MINING ENGINEERING

JANUARY 1951

In Two Sections—Section 1

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See P. 17

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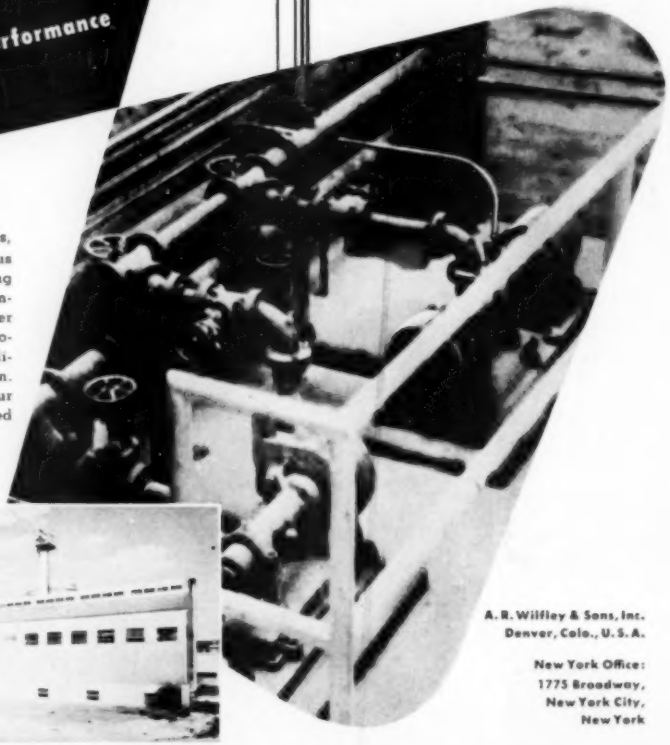

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Incorporating Mining and Metallurgy, Mining Technology and Coal Technology

VOL. 190 NO. 1 JANUARY, 1951

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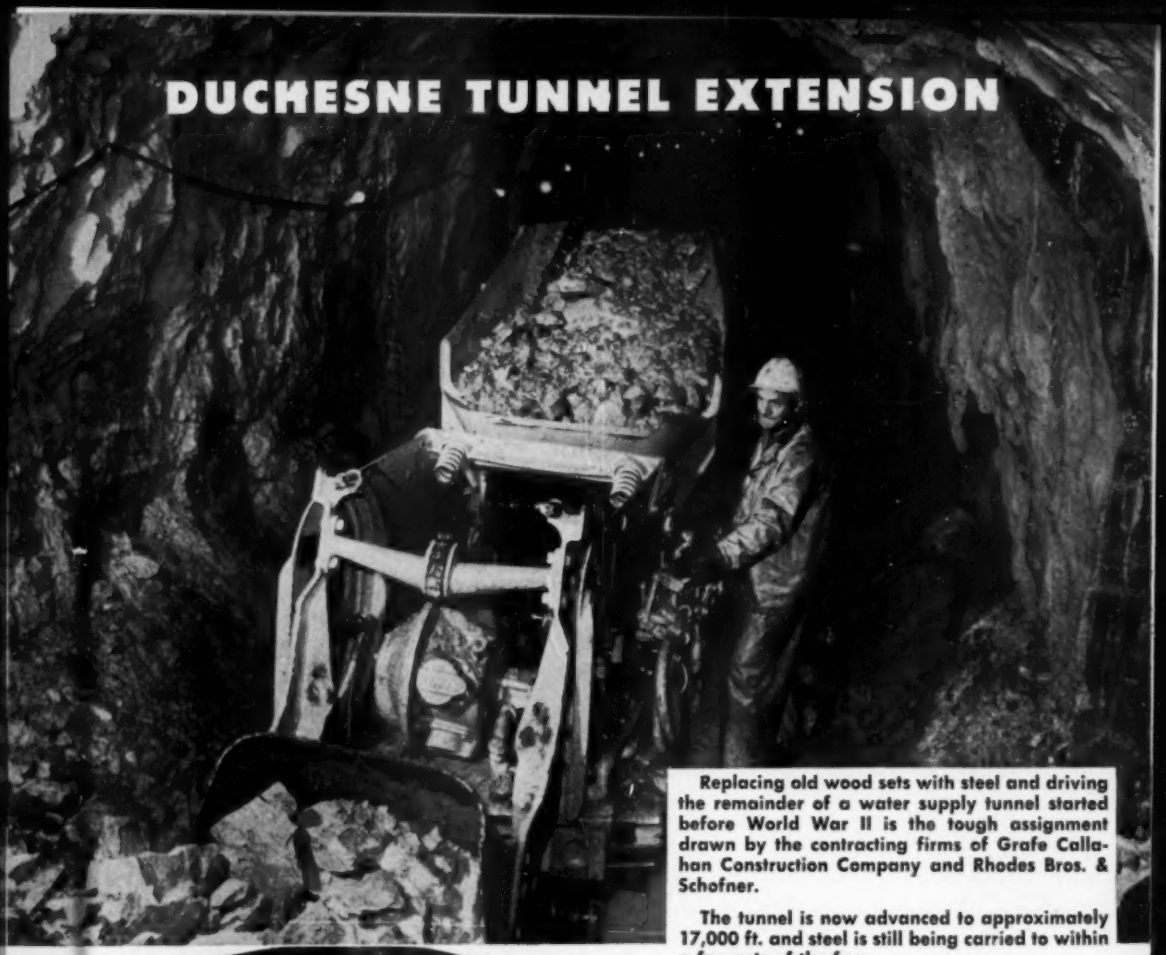

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A274

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(Continued on page 5)

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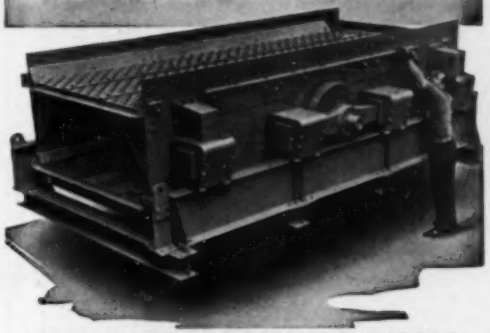

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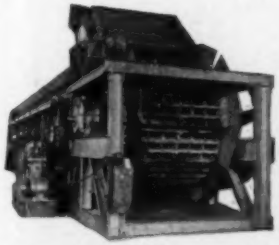

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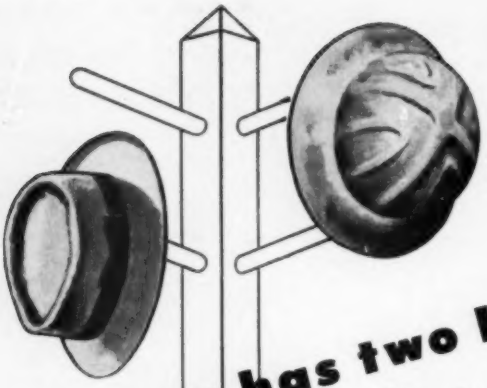

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Personnel Service

(Continued from page 3)

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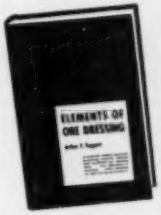

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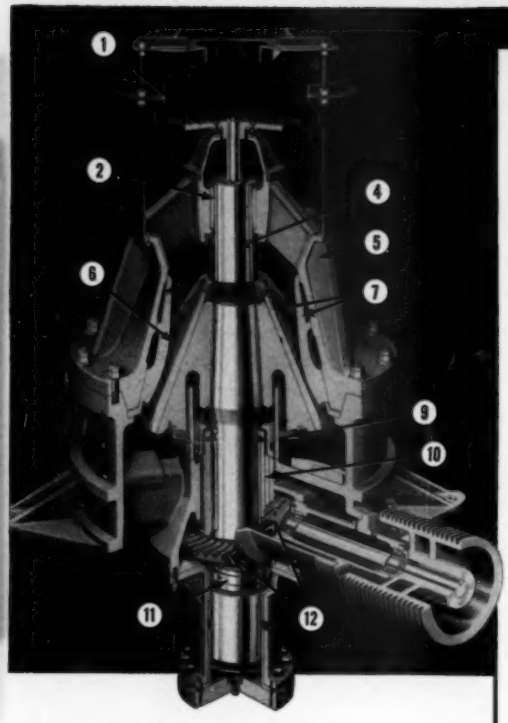

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Meet The Authors

E. N. Patty

W. Mitchell, Jr.

Will Mitchell, Jr. (P. 45), now director of Allis-Chalmers' noted basic industries research laboratory, has B.S. and M.S. degrees from the Montana School of Mines, and has done graduate work at MIT. At first he taught chemistry at Montana, then he was assistant ventilation engineer at the Puget Sound Navy Yard, assistant mining engineer for Anaconda, and an instructor in mineral dressing at Lafayette College. At MIT he served as administrative assistant and research engineer in the mineral engineering laboratory. He was born in Butte, Mont., now lives in Milwaukee. Mr. Mitchell serves AIME as Secretary-Treasurer of the Minerals Beneficiation Div.

E. P. Patty (P. 27) who was for 13 years dean of the faculty and head of the school of mines at the University of Alaska, is now president and general manager of three gold dredging companies operating in Alaska and Canada. Mr. Patty was born in La Grande, Ore., attended Lincoln high school in Portland, and went to the University of Washington for his B.S. and E.M. degrees. He spent two years as a mining engineer with the Washington State Geological Survey, and a year managing the Black Rock mine in Northport, Wash. This is the second paper that AIME member Patty has presented before the Institute. In his spare time he enjoys fishing, gardening, and showing his large collection of home movies on Alaska, his travels, and various mining operations.

F. S. McNicholas (P. 33), consulting engineer for the Climax Molybdenum Co. and for the Mufulira Copper Mining Co., last wrote for us in November (*Caving and Drawing at Climax*, P. 1116). A complete biography appeared in that issue. We did, however, err in that biography. Mr. McNicholas was mine superintendent for Granby Consolidated for the greater part of the 10 years he was with the company, and not assistant mine superintendent, as we reported.

(*Continued on page 8*)

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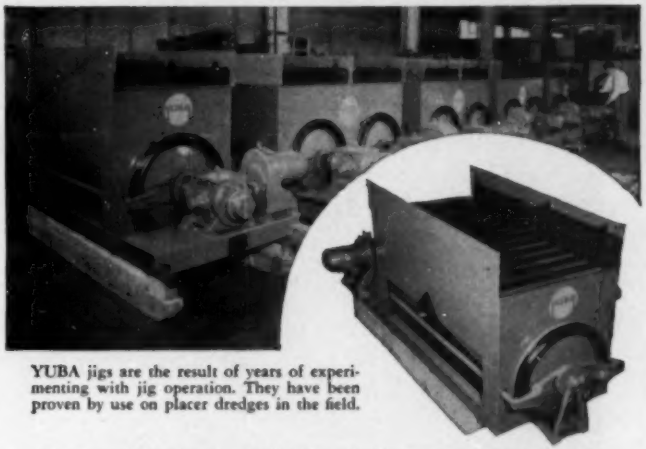

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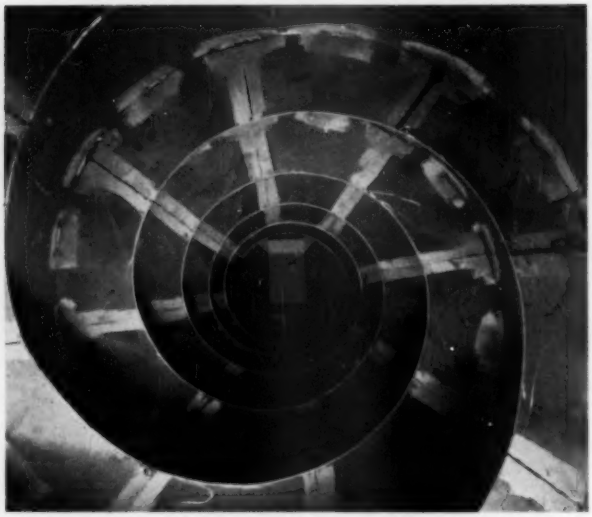

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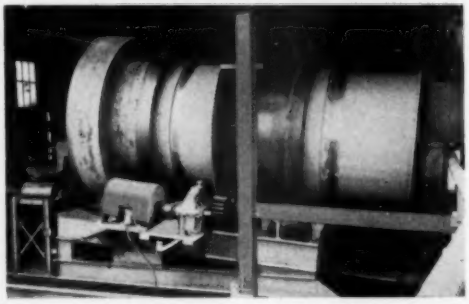

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Meet The Authors

(Continued from page 7)

R. H. Jahns

C. Sollenberger

R. H. Jahns (P. 50) is now serving as both senior geologist with the U.S. Geological Survey and as a professor of geology at the California Institute of Technology. He has been with USGS since 1937, and on the teaching staff of CIT for four years. Born in Los Angeles, he attended high school in Seattle, attended CIT and Northwestern University. He has B.S. and Ph.D. degrees from CIT, and an MS from Northwestern. Dr. Jahns has presented several other papers on the geology of pegmatites and pegmatite deposits before AIME. His hobbies include railroads, and especially railroad photography.

C. L. Sollenberger (P. 45), co-author with Messrs. Mitchell and Kirkland, is an ore testing engineer for the Allis-Chalmers Mfg. Co., has been at A-C's basic industries laboratories for the past two years. A graduate of Lehigh, with a degree in chemical engineering, he took his MS in metallurgy at MIT, later returned to work at MIT's mineral engineering laboratory in Watertown, Mass. Mr. Sollenberger was born in Carlisle, Pa., and attended high school there. He has been a Junior Member of AIME since 1947.

G. L. Simard (P. 39), group leader with the research division of the American Cyanamid Co. in Stamford, Conn., was formerly a research engineer for Battelle Memorial Institute. Dr. Simard attended Bates College and MIT, holds a Ph.D. degree.

D. J. Salley (P. 39) has been with American Cyanamid since 1936, and is now with that firm's research division in Stamford, Conn. He holds a B.A. from the University of North Dakota and a Ph.D. from Princeton. Dr. Salley was with the Bell Telephone Laboratories in 1928 and 1929, and was a Textile Foundation Fellow from 1932 and 1935.

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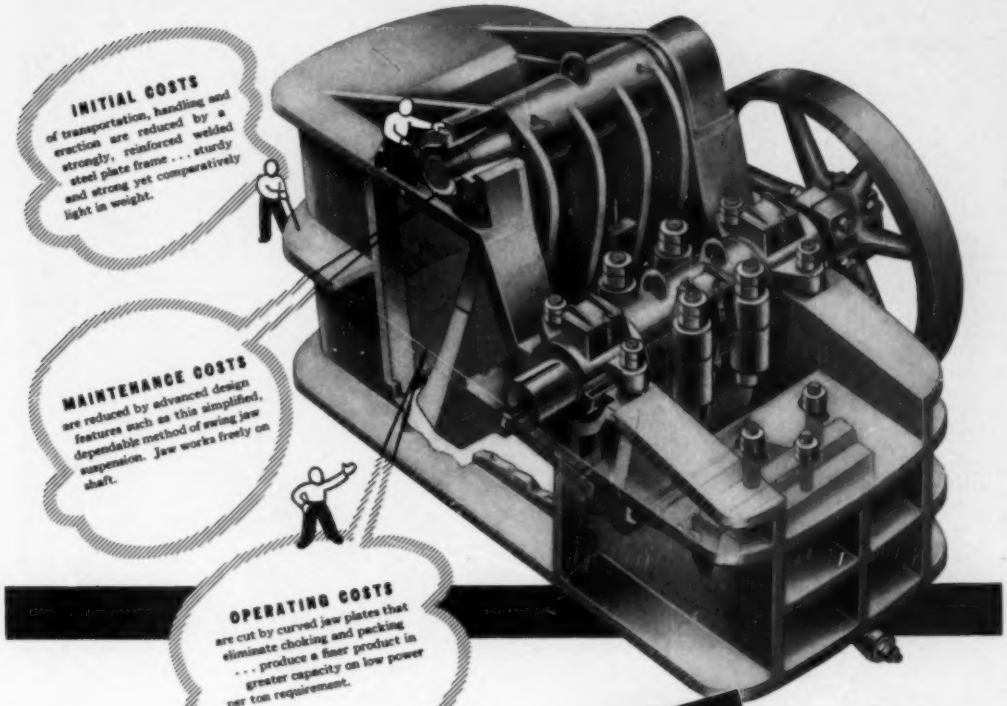

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Perlite, the lightweight aggregate replacing sand as an aggregate in plaster and concrete, is now processed in 75 plants throughout the country. Production in 1950 was 350 pct more than in 1949.

Additional discoveries of pitchblende have been made in highly sheared stringer zones in the footwall of the Sunshine vein proper on both the 3000 and 3100-ft levels of the Sunshine mine, Kellogg, Idaho. The first discovery was made on the 3700-ft level. Comprehensive development work is being undertaken to determine commercial possibilities of the ore between the 3000 and 3700-ft levels.

Uranium mines of Eastern Germany are shortly to be returned to German ownership. This is considered, by reliable sources, to mean that they have run out of uranium ore.

A new process for the recovery of cobalt from arsenic or nickel compounds has been developed through the pilot plant stage by the Chemical Construction Co. The process is to be applied to Blackbird, Idaho and Fredericktown, Mo. concentrates. Details of the process have not been announced.

Howe Sound Co. is negotiating with the Government to increase cobalt production capacity from 2 million to 3 million lb annually. For further details on the cobalt situation see "Cobalt" story in this issue.

200 million lb of increased aluminum capacity will be brought into production by Reynolds Metals Co. by 1952.

National Security Resources Board will recreate a Conservation Coordinating Committee such as operated during the last war. The Committee will initiate and develop broad unified programs of conservation in the mobilization effort.

HELPFUL FACTS ABOUT flotation

IN THIS 16-PAGE BOOKLET

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A Foreign Policy To Protect Our Resources

AGENCIES are being established in Washington to marshall raw materials and industry to build the machines of war. For the third time in less than a generation we are digging into nature's treasure trove for metals and fuels indispensable for war but equally indispensable to the peace of future generations of Americans.

Twice this nation has been enmeshed in global conflict by unheralded attack. Today we know the enemy. His philosophy has been spelled out by scholars; he admits his objectives. Communism is dedicated to the envelopment of the world—not today or tomorrow but through the years the iron curtain is being pushed outward. The Soviet Communists are an older regime than any we have fought before. The weapons of this enemy are manifest; in our history books, in Berlin, in the annals of our courts, and in the rice paddies of Korea. No global war will be precipitated by the Soviets until they believe victory is assured.

With this thoroughly understood, a foreign policy must be formulated. We submit the following: Continued and increased support of the Atlantic Pact, exporting all types of arms to Europe; supplying weapons and military advice to those who have the will and ability to resist Communist aggression wherever it occurs; and a continuous barrage of propaganda for the democratic way of life, as opposed to the Communistic police state, that will reach the people of Communistic countries. Added to this might well be greater support to a strengthening of the United Nations, with codified international law and an international police army that can give effect to United Nations decisions. Production of armaments for export will make our industrial plants ready for the supreme emergency if it should come. However, this sort of manufacturing for a protracted period would be a severe drain on our mineral supplies. This must be avoided by importing the raw materials for this export business and by the efficient exploitation of our own. To this end government support of mining in the form of loans, increased amortization rates, and subsidies may be necessary. It will no doubt also be necessary to restrain allocations of critical raw materials for nonessential uses.

The procurement and allocation agencies must be permitted to function in a nonpolitical climate and they must receive the cooperation of industry. The military program is well under way but the foreign policy framework has not been established.

This nation is committed to the defense of the democratic world against Communist aggression. We have the right to demand that a foreign policy designed to thwart the known objectives of the Soviet Union be established. In this supreme effort we are making, our manpower and our irreplaceable natural resources must not be wasted by vacillating foreign policies.

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Industrial Minerals and Rocks

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Fluorspar and Cryolite
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Chalk and Whiting
Strontium Minerals
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Washington By-line

OFFICIAL Washington is gravely concerned over the adverse military situation caused by the all-out intervention of the Chinese Communists in Korea. What the policies of the United Nations and the United States will be in the new crisis is unknown. However, Stuart Symington, chairman of the NSRB, set the general pattern for mobilization in the near future when he said that this country would move from a "light gray to a dark gray mobilization."

The implications of accelerated mobilization for the mining industry are particularly apparent with respect to the manpower situation. Revised policies recently initiated for the recall of reservists may be abrogated, in which case key personnel in the mining industry may face calls to active duty with insufficient consideration given to their indispensability to national defense in their present positions.

Mine Loans

According to authoritative sources, applications for loan contracts for exploration of new deposits of strategic and critical metals and minerals will not be acted upon until a little after the first of the year. The detailed plans covering loans and grants for exploration have not yet been worked out. At present the accent is on greatly stimulating the extraction of ores from producing deposits.

Application forms for direct government loans are now available and can be secured through the National Security Resources Board.

The Defense Minerals Administration is now prepared to process private loans which, if certified, will be guaranteed by the proper Federal Reserve Bank.

The NSRB recently issued a set of rules on the powers and duties of defense agencies and the RFC with regard to terms and conditions of loans for the purposes set forth in the Defense Production Act of 1950. These rules delegate to the Dept. of the Interior, in the case of mining loans, the power to determine: (1) the period of the loan (but not to exceed 20 years unless authorized by the chairman of the NSRB), (2) the amount of the loan, and (3) the time for and the method of loan repayment. The rules provide that in a case where collateral is insufficient the loan may be made on a partially secured or completely unsecured basis if "the managerial ability of the borrower, the potentiality of the profits of the enterprise and other relevant factors bearing on the ability of the borrower to repay which reasonably assure the repayment of the loan." Upon the receipt of a certificate from the Interior Dept., the RFC is "authorized and directed" to make the loan under the terms and conditions set forth in such certificate. In other words, approval by the Interior Dept. is tantamount to receipt of a loan.

Metals Limitation Orders

Restrictions on consumption of essential metals for national defense have been expanding at a rapid rate. Most of the critical metals have now been covered with varying degrees of intensity.

The NPA order on copper and copper base alloys limits the receipt of rated orders by producers of certain copper products to 5 to 25 pct of average monthly shipments of such products in the first six months of 1950. Copper producers and fabricators of certain products are granted lead time of 45 days.

The NPA order on zinc stipulates that the producers of zinc, zinc dust, and zinc oxide may refuse rated orders that are in excess of 10 pct of the producer's scheduled production of such items in total tonnage during that month. The zinc producer must schedule the order for shipment within the requested month.

An order on nickel limits the quarterly consumption of nickel for other than its use in fulfilling military contracts to 65 pct of the average consumption of those products in the first six months of 1950.

An amended order on aluminum eased the original 35 pct reduction in consumption of certain aluminum products by providing that the reduction should be only 20 pct in January 1951 and 25 pct in February 1951 of the average monthly use of such products for the first six months of 1950.

Restrictions on the use of cobalt were modified recently. A late November order cut the consumption of cobalt for December by 50 pct as compared to a 70 pct reduction ordered in an earlier directive.

All of the orders mentioned above and issued up until December 1 or a little after do not reflect the new international crisis posed by the Chinese menace.

Excess-Profits-Tax Exemption

The excess-profits-tax bill, HR 9827, was passed by the House on December 5. Domestic corporations engaged in the mining of certain strategic or critical minerals are exempt from an excess-profits tax to the extent that their net income is attributable to the mining of such minerals. This provision is the same as that in the World War II excess-profits-tax law except that certain minerals were added to this section by a House floor amendment. The following minerals come under the exemption: antimony, chromite, manganese, nickel, platinum, quicksilver, sheet mica, tantalum, tin, tungsten, vanadium, fluorspar, flake graphite, vermiculite, long fiber asbestos, chrysotile, beryl, cobalt, columbite, corundum, diamonds, kyanite, monazite, quartz crystal, and uranium. Upon the certification of the Defense Minerals Administration, other minerals deemed to be essential to the defense effort may be added to the list.

The House bill also includes the provisions of Section 735 of the prior law exempting the excess output of mines, above the normal rate, from an excess-profits tax. It was understood that this Section would remain in its present form until sometime in 1951 when it would be possible to review this provision in greater detail.

Steel Requirements for Mining

The Committee on Technology of the National Minerals Advisory Council recently submitted a report to the Dept. of the Interior on steel and maintenance, repair, and operating requirements for the mining industry for 1951. The data contained in the report were gathered by the American Mining Congress for the liaison committee appointed by the Bureau of Mines to coordinate the efforts of the National Bituminous Coal Advisory Council and the National Minerals Advisory Council. The report has not been made public but it is understood that emphasis was placed on the inadequacy of steel for mining machinery. It is also known that rated orders for steel to be used in the production of vitally-needed mining machinery were being refused for the fourth quarter of 1950 and that steel orders already placed with mills are being set back in the mills' schedules. Such handicaps to the production of essential metals and minerals should not be allowed to continue much longer.

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by John V. Beall

Editor

MINING ENGINEERING

BROMO Seltzer blue has gone to war. The blue of the Bromo Seltzer bottle is a product of cobalt, the Nation's No. 1 strategic metal. When the National Production Authority, on Nov. 21, 1950, ordered a 70 pct cutback in domestic use of cobalt, a stab of pain was felt in the hearts of the glass, porcelain enamel, steel, paint, pottery, and electrical and electronic equipment industries. The reason: A full-scale wartime economy with astronomical requirements for jet engines, high velocity projectiles, armor plate, electronic equipment, and the many products needed to win a war. There is little potential comfort for cobalt users in nonessential manufacturing, and the military will be pinched for optimum supplies of the metal. However, if proper measures are taken immediately and current plans are completed, in two years there will be far more cobalt available than ever before. When normal times return, cobalt supplies should be sufficient to meet peacetime military requirements, and greatly expanded domestic requirements that will result from new technological development for wartime but adaptable to peacetime products. The pres-

Drilling a vein rich in cobalt and silver preparatory to blasting in Silanco Mining & Refinery Co's. Temiskaming mine. Cobalt vein is shown by bloom in upper right hand corner.

ent crisis is a challenge to the mining and metallurgical industries.

The name cobalt, derived from the German *kobold* meaning goblin or something bad, was applied because the high arsenic content in the ore caused sores on miners' hands. Long before Christianity, cobalt was used as a pigment and to provide a hard, glossy surface to ceramics. Prior to World War I, cobalt consumption in the United States was low, and cobalt was imported from Canada as an oxide for the ceramic industry. Since 1922 the demand for the metal has increased steadily to about 8 million lb a year, as is shown in Table I, compiled from statistics in the Bureau of Mines *Minerals Yearbook, 1949*. Nearly all cobalt used in the United States is imported, and consequently these figures actually indicate consumption.

The ceramic industry is the oldest consumer of cobalt. It has been used classically to provide a blue color. It makes an underglaze or overglaze stain, which, when combined with the oxides of chromium, nickel, and manganese, yields all shades of blue or green. The blue of the Bromo Seltzer bottle is a familiar application of its use. It also counteracts the yellow stains from traces of iron in clays used for whiteware. About 1 lb of cobalt oxide per ton of dry body is used for this purpose. Cobalt oxide is used in frit or ground coats in porcelain enamelware because it enhances bonding at the steel-enamel interface. This consumption totaled about 424,000 lb in 1949, but only part of it is considered essential when wartime restrictions are in force.

Carbonates, hydroxides, acetates, and other salts of cobalt are used as driers in paints and varnishes. There are substitutes, but they are not effective. Much success has been achieved by using cobalt to supplement pasture deficiencies in cattle raising. Consumption of cobalt for these combined uses totaled 765,000 lb in 1949.

Imports of cobalt in all forms increased steadily in the two decades from 1920 to 1940. During the past decade, however, alloy and ore imports have varied and imports of oxides, sulphates, and other compounds have declined. Imports of the metal have increased phenomenally, the result of increased metallurgical use. The decline in imports of cobalt compounds is probably the result of their increased production in the United States from the ores and alloys imported. Table I illustrates these trends. Discrepancies between the figures for "Imported for Consumption" (Table I) and "Consumption by Uses" (Table II) is accounted for by consumer stocks and U. S. stockpiles. By the end of 1949 imports of the metal had reached over 5 million lb.

In 1941, it was first reported in the *Minerals Yearbook* that the largest single use of cobalt was in the production of Stellite-type alloys. Permanent magnet production now accounts for the greatest amount; and high-speed steels, hard-facing materials, and cemented carbides are large consumers. If figures on cobalt consumption are released in the coming months, it is likely that the refractory-type alloys used in gas turbines will be the largest single consumer. The impact of these alloys on cobalt supplies has not yet been fully felt.

Cobalt in high-speed steels improves cutting performance at high speeds and elevated temperatures. It is used in certain high-speed steels with tungsten, chromium, molybdenum, vanadium, and other alloying elements in amounts up to about 13 pct. Cobalt is desirable in many, and necessary in some of these high-speed alloys. Only a small percentage of high-speed steel grades, however, contains cobalt, 283,496 lb being used in 1949.

The big consumer of cobalt is in refractory-type alloy production, with 1,238,083 lb going into these products in 1949. Stellite is the trademark of the Haynes Stellite Co. for a group of alloys, used as refractory materials, cutting tools, and hard facing material. These contain 45 to 55 pct cobalt, 30 to 35 pct chromium, and 12 to 17 pct tungsten. Cemented carbides accounted for 118,522 lb of cobalt in 1949. Tungsten and other cemented carbides are sintered with 3 to 20 pct metallic cobalt as a binder. Both the Stellite alloys and cemented carbides are highly efficient heavy-duty metal cutting and metal die materials. They are used for drawing, extruding, and cutting all manner of substances from hard metals and ceramic ware to soft plastic materials, wood, and carbon.

Substitution of vanadium, tantalum, and boron for cobalt in high-speed steels are possibilities; and nickel can be used as a binder in cemented carbides but with impairment of efficiency. J. P. Gill, Vanadium Alloys Steel Corp., Latrobe, Pa., in 1942 suggested various substitutions for cobalt in high-speed steels. Four German patents have been reported for various hard cast-alloys which eliminate cobalt. These contain: W-B-Si; Mo-Mn-Cr; W-Fe-Ti-C; W-Fe-Ti-Ce. Substitutions, however, are not

Table I—Cobalt Imported for Consumption in the United States, Lb[1]

Year	Alloy	Ore	Metal	Oxide	Sulphate and Other Compounds	Gross Weight	Cobalt Content (Estimated)
1923 to 1924 Avg		43,753	172,296	242,639	23,220	481,908	354,500
1925 to 1929 Avg		158,315	467,080	366,045	47,758	1,039,198	752,200
1930 to 1934 Avg	[2]	323,072	307,232	374,091	67,347	1,071,742	655,200
1935 to 1939 Avg	[2]	621,487	1,115,589	653,486	60,466	2,453,368	1,679,000
1940	7,843,828	2,653,891	130,321	756,759	11,468	11,396,267	4,200,000
1941[3]	9,970,589	2,443,725	554,030	38,002	4,980	13,011,326[4]	4,328,000[5]
1942	10,313,867	834,797	148,304	——	200	11,297,168	4,280,000
1943	10,110,879	10,556,042	266,670	59,928	56	20,992,575	5,626,000
1944	8,500,516	473,529	73,088	225,809	115	9,272,857	3,796,000
1945	8,379,145	859,940	946,475	120,872	224	10,324,456	4,615,000
1946	1,848,595	627,787	1,935,589	1,074,630	350	5,316,944	3,451,000
1947	3,751,453	751,438	6,035,153	752,150	530	11,290,723	8,206,000
1948	4,879,413	6,167,545	5,266,521	790,300	1,374	19,105,153	8,821,000
1949	3,691,051	109,009	5,588,327	360,318	359	9,749,064	7,458,000
First ¾ 1950	2,888,178	116,757	5,283,294	629,999	1,414	8,919,642	6,610,695[6]

[1] Compiled from Bureau of Mines *Minerals Yearbook*.
[2] A small amount of alloy was imported in 1934 and 1935.
[3] In addition, 4,796,000 lb of Burmese speiss containing 335,721 lb of cobalt were imported.

always the answer because often the substitutions involve using one strategic metal for another. Practically speaking, none can be counted on to conserve cobalt.

Taking the lead in the postwar demand for cobalt are permanent magnets and magnet steels, production of which reached an all-time high in the first nine months of 1950, consuming 2,081,000 lb. The development of Alnico permanent magnet alloys accounted for a large percentage of this demand. Alnico 7, developed in 1949, was designed for high demagnetization force applications particularly in motors, generators and air-gap devices. Alnico 5 DG is used in loudspeakers and high field strength magnetic assemblies such as holding-magnet assemblies, magnetic chucks, and radar assemblies. It is also used extensively in television, and the 70 pct curtailment order has given that industry more to worry about than the controversy over color television. Other uses of the magnet are in communication, magnetic separators, and novelty fields. For instance, General Electric uses Alnico magnets on its new refrigerator doors so the harrassed housewife can close the door with her hip while holding a baby in one hand and a bottle in the other. This use is sure to go with the new restrictions.

The property of cobalt alloy magnets that makes them so attractive is that they are highly efficient. More magnetic energy may be stored in a unit volume of a cobalt magnet alloy than in any other alloy. Permanent magnet steels contain up to 40 pct cobalt with about 0.9 pct carbon. They may be cast, annealed, forged, machined, or otherwise shaped, and subsequently hardened and magnetized. It is almost impossible to find a substitute for cobalt in magnet steels, although some conservation may be achieved by using various complex steels having less cobalt, such as chromium-cobalt alloy.

Precipitation-hardening magnet alloys differ from the magnet steels in that they contain little or no carbon, are not magnetic *as-cast*, and are given the desired properties by precipitation of one or more of the alloying constituents from a solid solution under controlled conditions. These magnet materials can be prepared by powder metallurgical techniques. Although they represent the highest performance magnetic alloys known, they often are difficult to forge or machine. The precipitation-hardened magnet alloys offer several possibilities in substitutes for cobalt. Molybdenum and tungsten-chromium-molybdenum dispersion-hardened alloys can be substituted for low-cobalt magnets.

Chromium-cobalt magnet steels may be replaced by dispersion-hardened molybdenum or tungsten-chromium-molybdenum alloys. Tungsten-cobalt steels containing 17 pct cobalt may be replaced by precipitation-hardened alloys containing 6 to 10 pct cobalt and 16 to 19 pct molybdenum.

When mechanical forming is necessary, there are no substitutes for the 35 to 41 pct cobalt alloys. However, where the product can be cast to shape, Mishima-type alloys containing iron-nickel-aluminum, may be substituted for most of the above cobalt alloys. There is, however, no known substitute for cobalt in the extremely high performance alloys of the new Alnico types.

Table III shows the relative efficiency of cobalt in some magnetic alloys. In the table, BrHc is the product: Retentive Power of the Magnet times the Coercive Force to Magnetize it; and is roughly proportional to the magnetic energy that can be stored in a unit volume of magnetic material. The product BrHc divided by the Percent of Cobalt is used in the table to compare the relative efficiencies of the use of cobalt in the various alloys.

The reason for treating the subject of magnetic steels at length is because they are important in military mobilization. To take one example, magnets used in instruments of guided missiles frequently must be contained in an instrument case that is often no more than 1-in. diam. Where weight and space are vital, the cobalt-alloy magnet materials are indispensable.

The past decade has seen the development of the so-called refractory types, or high temperature alloys capable of maintaining strength at operating temperatures of 1600°F or over, and capable of resisting creep, wear, corrosion, and erosion. Over 40

Table II—Cobalt Consumed in the United States by Uses, Lb of Cobalt[1]

Use	1943	1944	1945	1946	1947	1948	1949	1950 (9M)
Metallic:								
High speed steel	398,736	361,437	335,059	224,649	223,148	289,391	283,496	165,166
Other steel	29,604	36,865	25,566	201,949	87,710	132,803	162,638	177,590
Permanent—magnet alloys	1,032,374	1,322,097	1,105,880	1,463,539	1,106,147	1,353,371	1,194,920	2,081,050
Soft—magnetic alloys							42,980	7,576
Cast Cobalt-Chromium-Tungsten-Molybdenum Alloys	2,041,409[2]	1,757,673[2]	1,387,706	526,504	941,087	1,196,608	1,238,083	1,307,180
Alloy hard facing rods and materials	70,068	89,061	31,293	83,874	71,548	116,313	82,965	110,546
Cemented carbides			206,189	45,100	51,917	65,314	116,522	87,067
Other metallic	186,664	223,782	318,907	81,969	99,476	115,255	116,344	150,491
Total metallic	3,759,855	3,809,915	3,410,750	2,597,003	2,491,039	3,269,055	3,239,933	4,096,696
Nonmetallic (exclusive of salts and driers)								
Ground-coat frit	80,504	116,884	173,428	412,768	607,316	613,745	424,051	430,113
Pigments	39,384	53,654	100,789	170,602	207,928	232,725	188,606	143,207
Other nonmetallic	35,415	17,459	25,920	39,596	51,439	66,659	84,336	33,501
Total nonmetallic	155,303	187,997	300,137	623,034	866,683	913,129	696,993	606,821
Salts and driers lacquers, paints, inks, pigments, enamels, glazes, feed, electroplating, etc. (estimate)				885,000	797,000	818,000	765,000	735,000
Grand Total	3,915,158	3,988,912	3,710,887	4,105,037	4,154,722	5,019,224	4,701,926	5,448,517

[1] Compiled from Bureau of Mines *Minerals Yearbook*.
[2] Includes cemented carbides.

such alloys have been developed and at least 13 contain cobalt ranging from 13 to 66 pct. The alloys were designed for components of gas turbines, jet aircraft engines, and turbo-superchargers. Table IV shows the analyses of various of these high-temperature alloys. Vastly greater amounts of cobalt will be required by expansion of military requirements, but as stated earlier the impact of consumption for this work has not been realized fully.

From the standpoint of military mobilization, cobalt is important, and in Washington circles is considered about the "hottest" strategic metal on the list. Cobalt used in dies and cutting tools for defense orders will receive priority. Hard surfacing materials and alloys containing cobalt for high temperature and erosion resistance will be critical. Uses of cobalt for surgical and dental instruments, and as a powder metal binder for electronic instrument parts will receive priority. In magnetic steels, cobalt will be essential where maximum efficiency and size and weight limitations are factors. All of these requirements will seriously strain available supplies; but restricting nonessential consumer use, substituting where possible, and developing cobalt ore bodies in North America and purchasing from abroad will ease the situation.

As shown in Table I, the annual requirements are about 8 million lb, prior to the present emergency. If appropriate steps are taken at least half of this amount can be supplied by commercial recovery of the metal from ores on the North American continent. However, before going into this hypothesis it is desirable to review the present world sources.

Cobalt Sources

Despite the fact that cobalt is produced in many countries, Belgian Congo, Northern Rhodesia, United States, Canada, and French Morroco account for 95 pct of the production. Iron pyrites contain some cobalt, often only 0.05 pct, and it has been recovered from deposits in Finland, Germany, Greece, Italy, Norway, Spain, and Sweden. Germany obtained cobalt from iron pyrites during World War II.

The world's largest producer is the Belgian Congo. One company, the Union Minière du Haut Katanga, accounts for all of the production from this country. This black oxide cobalt is a byproduct of the mining of oxidized copper ores. The property is operating at peak production since the expansion of concentrating and refining capacity and the reopening of the Ruashi mine near Elisabethville. Production statistics for 1950 are not available, but in 1949 it set a record production of 4350 metric tons of contained cobalt. The property of the Union Minière in the Congo consists of the Kabolela mine and cobalt concentrating plant. This mine carries higher values in cobalt than in copper but is past its peak production. At Kamoto there is another cobalt mine and ore-treatment plant. Mines and concentrators are also operated at Kolwezi and Kipushi, but these are cobalt-bearing high-copper ores. At Jadotville, the company has six single-phase electric furnaces for smelting cobalt-bearing ores and slags. This plant has a capacity of about 300 tons a month. The cobaltiferous red alloy from the electric furnaces is refined in two rotary furnaces; and the resultant crude cobalt, which is a white metal alloy containing about 43 pct Co and impurities of Cu and Fe, is shipped to plants at Niagara Falls, N. Y. and Oolen, Belgium. These refineries process the alloy to metal, oxide, salts, and driers. The plant at Niagara Falls is operated by the Electro Metallurgical Corp., a subsidiary of the Union Carbide & Carbon Corp., but is owned by the African Metals Corp., the selling company in the United States for Union Minière.

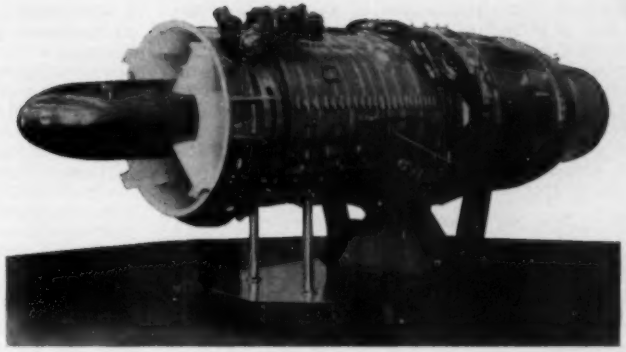
The cobalt in the solutions used in the electrolytic copper plant at Jadotville is recovered by precipitation. The precipitates are treated by electrolysis in a refining plant capable of producing about 225 tons a month of high purity granules. The total annual refining capacity of the company's plants at Niagara Falls, Oolen, and Jadotville is about 7500 metric tons. Converted to pounds, this is equivalent to 9,587,400 lb of contained cobalt, of which 7,490,188 lb of oxide, metal, and alloy were shipped to the U. S. in 1949. For the first nine months of 1950, 6,597,425 lb of contained cobalt has been shipped to the United States. Since the Union Minière began production in the Congo in 1924, 36,706 metric tons have been produced. The first year's production, 1924, amounted to 273 metric tons. Between 1936 and 1937 production was more than doubled. Since 1937, when the output was 1500 tons, production increased steadily to the record year of 1949.

Prior to November 1950, there always have been adequate supplies to meet world demand. Most of this supply has been met by a single company, Union Minière. Since Apr. 1, 1949, the price of cobalt has remained steady at $1.80 per lb for 97 to 99 pct pure in kegs of 550 lb delivered east of Chicago. The United States has had excellent relations in regard to allocation of metal and price in dealing with Union Minière. There is no tariff on cobalt. Producing at the rate of 4000 metric tons annually, the company has sufficient reserves for 40 to 50 years proved and expects that additional development work will disclose more ore. The high cobalt ores are diminishing and the bulk of the shipped product will shift from the white metal alloy recovered from these ores to the granules recovered as precipitates from the solutions in electrolytic refining of high copper ores.

Recovery of cobalt from the cobaltiferous copper ores of Katanga is not complicated by the presence of nickel or arsenic. The cobalt-bearing copper concentrates are reduced in the blast furnace and part of the cobalt is in the slag, which is treated further. The cobalt that remains with the copper is recovered by precipitation. Fines from cobalt concentrates and slags are first sintered in a Dwight-Lloyd

Table III—Relative Efficiency of Cobalt Utilization in Various Cobalt Magnetic Alloys

Type or Class of Alloy	Co, Pct	Brlle x 10⁻⁴ Gauss-Oersteds	Brlle x 10⁻⁴/ Pct Co
Commercial steel, low cobalt	8.5	0.9	106
Commercial steel, Co-Cr	16.0	1.44	90
Commercial steel, Co-W	17.0	1.49	88
Commercial steel, 35 to 41 pct Co	38.0	2.28	60
Mishima, 17 pct, Ni; 9 pct Al	5.0	1.51	300
Alnico, 20 pct Ni; 12 pct Al	5.0	3.57	712
24 pct Co, 28 pct W dispersion hardened	24.0	1.42	59
6 pct Co, 19 pct Mo dispersion-hardened	6.0	2.04	340
12 pct Co, 19 pct Mo dispersion-hardened	12.0	2.52	210
Honda new KS, 16 pct Ni; 11 pct Ti	28.0	6.24	222
Fe_2O_3 + $CoFe_2O_4$; 1:1 by weight	12.8	1.35	106



Cobalt is an essential element in the production of jet aircraft engines, such as this British Armstrong Siddeley Sapphire, a 7200 lb static thrust turbo-jet engine.

sintering machine before charging into electric furnaces. Lime and nut coke are added to the charge. The metal tapped from the furnaces separates into two layers. The lower one is crude copper containing cobalt, which is treated in the rotary furnaces of the copper refinery. The upper layer is crude cobalt alloy. It is cast into ingots or granulated by pouring on a flat jet of cold water. The granules or ingots are sent to the United States or Belgium.

Rhokana Corp., in Northern Rhodesia, is the second largest producer of cobalt. The first significant production was 1,976,000 lb of alloy containing 1,018,000 lb of cobalt in 1934. In the years 1938, 1940, and 1942 to 1944 production of cobalt exceeded 2 million lb and in 1939, the record year, it was over 3 million lb. Production has declined since 1944 so that in 1949, 2,342,000 lb of alloy containing 886,000 lb of cobalt were produced. Production from Rhokana is consumed in the United Kingdom.

The cobalt-bearing copper minerals at the Nkana mine of Rhokana are almost entirely sulphides, with an average cobalt content of about 0.15 pct in the form of carrollite. Two concentrates are made, one high in copper and low in cobalt and the other high in cobalt and low in copper. These are treated separately in reverberatory furnaces. A sulphide matte containing copper, iron, cobalt, and sulphur is formed in the bottom of the furnace, while a slag layer is formed on top. The slag is waste and the matte goes to the converters. In the converters, silica is added to form a fluid slag. Cobalt oxidizes before the copper and is removed as a silicate in the slag together with iron and copper as oxides and sulphides and silicates of aluminum, calcium, and magnesium. The slag is granulated and sent to the electric furnace where an alloy containing copper, iron, and cobalt is produced. The alloy is granulated and is ready for shipment to the refinery. Increasing iron content of the ore encountered in deeper mining has necessitated retreatment of considerable quantities of alloy in the converters.

Laboratory and pilot-plant investigations on the production of electrolytic cobalt from a flotation concentrate have been in progress. As a consequence of these studies and a $550,000 grant from the Economic Cooperation Administration for plant expansion, it is expected that Rhokana production will be increased substantially in 1951.

French Morocco sometimes ranks third in the world production of cobalt. La Société Minière de Bou-Azzer et du Graara, Casablanca, is the sole producer. The Bou-Azzer ore body outcrops in pre-Cambrian strata, which form the substructure of the Anti-Atlas Mountains. The ore is found in small pockets but it is believed that there are larger ones. The cobalt minerals are smaltite and skutterudite. The average cobalt content is 11 to 12 pct, but there are other commercial deposits having only 0.75 to 1.25 pct. Gold is also recovered from the ore, in amounts of 5 to 30 g per ton. The ore also contains nickel and silver. Concentrates are shipped to Belgium for processing to oxide and

Table IV—Analyses of Cobalt High-Temperature Alloys, Pct

Alloy Designation	Co	C	Mn	Si	Cr	Ni	Mo	W	Fe	Other
Low Carbon N-155 Multimet	20.00	0.15	1.00	0.50	20.00	20.00	3.00	2.00	Bal.	N-0.14, Cb-1.00
Low Carbon N-155	13.00	0.15	1.50	0.50	16.00	15.00	3.00	2.00	Bal.	N-0.14, Cb-1.00
S-590	20.00	0.42	1.12	0.57	20.00	20.00	4.00	4.00	Bal.	Cb-4.00
S-816	44.00	0.40	0.62	0.57	20.00	20.00	4.00	4.00	4.00	Cb-4.00
K-42-B	22.00	0.05	0.70	0.40	18.00	43.00			13.00	Ti-2.00, Al-0.60
Refractalloy 26	20.00	0.03	0.70	0.65	18.00	37.00	3.00		Bal.	Ti-3.00, Al-0.30
Refractalloy 70	30.00	0.04	2.0	0.30	20.00	21.00	8.00	4.20	14.00	
Refractalloy 80	30.00	0.10	0.60	0.70	20.00	20.00	10.00	5.00	14.00	
Hastelloy B	29.00	0.05	0.60	0.25		65.00			4.50	
Vitallium	64.00	0.30	0.30	0.25	27.00	2.00	5.00		2.00	
Haynes Stellite 23	65.00	0.38	0.30	0.80	25.00	2.00		8.00	1.00	
Haynes Stellite 27	30.00	0.42	0.50	0.40	26.00	36.00	6.00		1.00	
Haynes Stellite 30	51.00	0.42	0.50	0.40	26.00	15.00	6.00		1.00	
Haynes Stellite 31	54.00	0.50	0.60	0.70	25.00	10.00		7.50	1.00	
X-40	54.50	0.45			26.50	11.00		7.25	0.60	
X-50	40.00	0.40	0.60	0.50	23.00	20.00		12.00	3.00	

speiss, which are exported to France where the speiss is refined to metal. After the German occupation of Morocco in World War II, the ore was shipped to the United States. ECA has made a contract to purchase 7200 metric tons of ore from French Morocco to be delivered by November 1952. This ore averages 12 pct cobalt and will be refined at Deloro, Ont. Production at Bou-Azzer began in 1935 and in 1938, 612,712 lb of cobalt were produced. In 1949, production declined to 460,636 lb.

Canadian Production

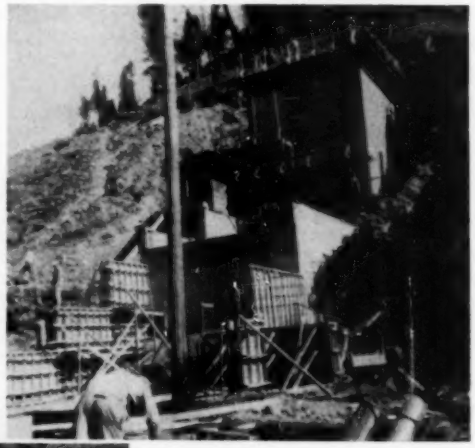
Canada has a history of cobalt production going back to 1904 when production from the fabulous Cobalt, Ontario, silver mines began. For three successive years, 1908 to 1910, production of cobalt metal exceeded 2 million lb; but it was not the cobalt that made this district prominent. Between 1904 and 1922, $300 million worth of silver was produced, which is the greatest production from one camp in such a short time in the world's history of silver mining. Today, the Cobalt district is being revived because of new silver discoveries in the old producing formation. Production of cobalt is tied up closely with the production of silver from these mines. High grade silver ores bearing cobalt are mined, but high cobalt ores with values in silver also can be produced. The problem is an economic one, because the miner does not get paid for the cobalt in the high silver ores and the price for cobalt, 80 to 85c per lb to the miner, is not high enough for him to mine the high cobalt ores. If the mining companies received $1.25 per lb for their cobalt or received payment for 85 pct of the contained cobalt in the silver concentrates, more of the cobalt-bearing ores would be mined.

At present, two mines are producing and milling ore, the Silver Miller Mines and the Cobalt Load Silver Mines; and 10 other properties are in the development stage. Deposits are fault fissure veins 4 to 5 in. wide, a few hundred ft long and usually less than 300 ft deep. Cobalt in the ore runs as high as 9 to 10 pct in the form of cobalt arsenides, cobaltite, and safflorite. Arsenopyrite and niccolite are also contained in the ore.

One of the main factors that held back recovery of cobalt was the absence of a smelter and refinery in the district. Concentrates were shipped to Deloro Smelting & Refining Co., Ont. There was not sufficient ore during World War II to keep the smelter operating and the smelter changed over to treat African ores. Concentrates from Ontario were stockpiled until a sufficient amount was accumulated to warrant operation of the smelter. In 1945, Silanco Mining & Smelting Corp. began construction of a smelter 1 mile south of Gillies, which was to have a daily capacity of 10 tons of concentrates, using electric furnaces for smelting and chemical treatment for separation of the copper, nickel, arsenic, and silver in the ores. The successor to Silanco, the Cobalt Chemical & Refinery Co. completed the smelter in 1949, but it burned down shortly afterwards. It is reported that the New Cross Chemical Co. plans to construct a plant adjacent to the smelter of Cobalt Chemical to produce salts, oxides, and other cobalt products.

As the situation now stands, the silver mines are perforce developing cobalt-bearing ore bodies, and there are probably considerable amounts of cobalt-bearing rock on the surface in mill tailings and waste piles. It is planned to reconstruct the burned out smelter. Cobalt Chemical & Refinery Co., has had a purchase order for $2,880,000 worth of cobalt from the United States Government since Dec. 5, 1949. One of the major difficulties in meeting the obligations of this contract has been in getting skilled technical assistance for solving the complicated metallurgy of recovering cobalt from these nickel-arsenic bearing ores. It is probable that 750,000 lb of cobalt could be produced annually from this area under proper conditions. These possibilities are being kept alive, receiving consideration in Washington circles, and a firm of well-known consultants has been retained to study production problems.

International Nickel Co. of Canada, Ltd. continued the recovery of cobalt as oxide at Port Colborne, Ont., refinery. This cobalt comes from the nickel-copper ores of the Sudbury district. In 1948, 15 short tons of oxide were produced and went to the United Kingdom. Expanded production is expected in 1952.



Work in progress at the Howe Sound Co's. Blackbird mine, Forney, Idaho. At left, the campsite in June 1947, and above, a recent picture of the coarse crushing plant under construction.

Falconbridge Nickel Mines, Ltd. continued work on a plant at its nickel refinery at Kristiansand, Norway, to produce cobalt oxide and electrolytic cobalt. This cobalt will be recovered from the matte produced from Sudbury nickel-copper ores. The plant is expected to be in operation in 1951.

Cobalt produced in Canada in 1949 amounted to 613,000 lb, as compared to 1,544,852 lb in 1948. The major portion of the 1948 output came from accumulations during the war years rather than from ore mined in 1948. Most of this production went to the United States.

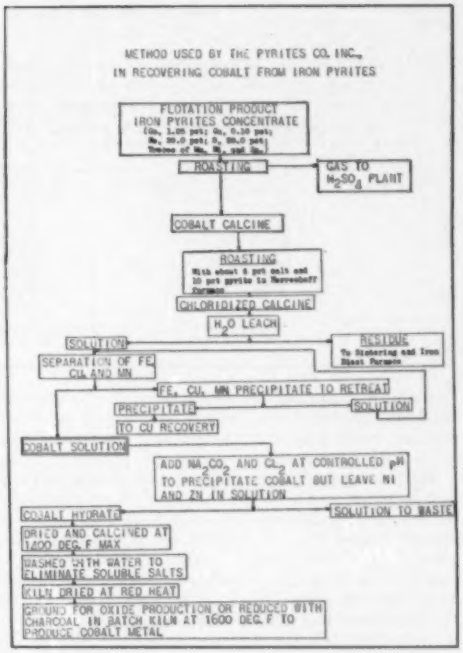
A cobalt-bearing ore body in Uganda, British East Africa, carrying 0.3 pct cobalt, 1½ pct copper and no nickel or arsenic, is reportedly under study by the interests of Ventures Ltd. and Rio Tinto Co. Ltd.

Burma production of cobalt was important to the United States in 1943 when 206,724 lb of speiss, which contained 7 pct cobalt, 30 pct nickel, 40 pct arsenic, and 10 pct copper, were imported. This speiss is still in the Government stockpile.

In 1948, the U.S. Government gave a contract to the Reduction & Refining Co. of Kenilworth, N. J. to recover cobalt from various ores and residues in the stockpile. Other countries have produced small quantities of cobalt, including Australia, New Caledonia, Bolivia, Chile, Finland, Italy, Japan and Sweden.

Despite the fact that the United States is the largest consumer in the world, only a small amount of cobalt is produced here. Bethlehem Steel Co. produced 673,773 lb in 1949. The cobalt comes from the iron pyrites that accompany the magnetite mined at Cornwall, Pa. The cobalt bearing material, averaging 1.3 pct cobalt in 1949, is shipped to the Pyrites Co., Wilmington, Del., where it is processed to metal and other cobalt products. Details of this process are shown in Table V. For the past 10 years this has been the only steady source of cobalt production in the United States. The Sullivan Mining Co., Kellogg, Idaho, has been recovering cobalt from the residues of its electrolytic zinc plant, but no shipments were made. In 1949, 10,000 lb were recovered.

Other companies and areas reporting small production of cobalt or development work on cobalt ore bodies include Eastern Magnesia Talc Co., Inc., Burlington, Vt., (recovered from talc); Cobalt Gold Mining Co., Boulder, Colo., (development work); Goodsprings district, Clark County, Nev. (development work on cobalt-copper ore body); property near Fort Thomas, Ariz. (small production); and Oregon, which has been credited with spasmodic shipments of arsenical ore containing cobalt, gold, and silver. A promising source of cobalt is from the properties of the St. Louis Smelting & Refining Co., near Fredericktown, Madison County, Mo. The company is a subsidiary of the National Lead Co. Prior to 1921 sporadic production on a small scale was achieved from the complex ore body which contains copper, lead, iron, and nickel, in addition to cobalt. Total output from Missouri did not exceed 700,000 lb. Large sums of money were spent on the development of metallurgical processing of the ore, but no process proved entirely satisfactory. In 1942 development of the ore bodies was undertaken and the U. S. Bureau of Mines made studies on concentrating the ore. Fairly satisfactory results were obtained by selective flotation of two concentrates, copper-lead and cobalt-nickel. Construction of a mill was started in 1943 and production began in July, 1944. A flotation process was used to make a lead concentrate,


METHOD USED BY THE PYRITES CO. INC.,
IN RECOVERING COBALT FROM IRON PYRITES
FLOTATION PRODUCT
IRON PYRITES CONCENTRATE
ROASTING
GAS TO H_2SO_4 PLANT
COBALT CALCINE
ROASTING
CHLORIDIZED CALCINE
H_2O LEACH
SOLUTION
RESIDUE
SEPARATION OF FE, CU, AND MN
FE, CU, MN PRECIPITATE TO RETREAT
SOLUTION
PRECIPITATE
TO CU RECOVERY
COBALT SOLUTION
ADD NA_2CO_2 AND CL_2 AT CONTROLLED pH TO PRECIPITATE COBALT BUT LEAVE NI AND ZN IN SOLUTION
COBALT HYDRATE
SOLUTION TO WASTE
DRIED AND CALCINED AT 1400 DEG. F MAX
WASHED WITH WATER TO ELIMINATE SOLUBLE SALTS
KILN DRIED AT RED HEAT
GROUND FOR OXIDE PRODUCTION OR REDUCED WITH CHARCOAL IN BATCH KILN AT 1600 DEG. F TO PRODUCE COBALT METAL

a copper concentrate containing some lead, and a nickel-cobalt concentrate. The roasted nickel-cobalt concentrate contained about 19 pct nickel and 12 pct cobalt. Some of this concentrate was shipped to the Ferro Enamel Supply Co. of Cleveland, for chemical recovery of separate oxide products of nickel and cobalt. In addition to the roasted concentrate, some ore and calcines were produced and shipped to Pyrites Co., at Wilmington. Some commercial production of cobalt was achieved from 1944 to 1946. Production of cobalt was suspended in September, 1946, but studies on the recovery of cobalt have been continued. Recently, concentrates of cobalt-nickel sulphides have been stockpiled, as the Fredericktown ore body has been worked for copper and lead.

The Chemical Construction Co., a subsidiary of American Cyanamid, has developed a process through the pilot plant stage that will treat the Fredericktown flotation concentrates and carry on through the production of the metal. This process has not been disclosed yet, but will probably be announced in the spring of 1951. The Chemical Construction

Table V—Sample Analyses of The Pyrites Co., Cobalt Products

Cobalt Metal Element	Range, Pct
Co	98.50 to 99.00
Cu	0.02 to 0.06
S	0.01 to 0.04
Fe	0.06 to 0.16
Ni	0.15 to 0.40
Mn	0.04 to 0.06
CaO	0.10 to 0.40
C	Less than 0.10

Cobalt Oxide Element	Range, Pct
Co	70.00 to 71.00
Cu	0.08 to 0.15
S	0.04 to 0.18
Fe	0.07 to 0.10
Ni	0.10 to 0.50
Mn	0.02 to 0.10
CaO	0.97 to 0.35

Screen	Pct
On 65 Mesh	0.0
On 100 Mesh	0.5
On 150 Mesh	1.0
On 200 Mesh	10.0

Table VII. Refiners or Processors of Cobalt in the United States, 1950

Refiner or Processor	Location of Plant	Cobalt Product[1] Made	Cobalt Raw Material[1] Used
Advance Solvents & Chemical Corp.	New York	F	B
African Metals Corp.	Niagara Falls	A, B, C, E	G
Ceramic Color & Chemical Mfg. Co.	New Brighton, Pa.	C, D, E	B
W. S. Chase Driers & Chemical Div., Industrial Lining Engineers, Inc.	Pittsburgh	F	D
Ferro Chemical Corp.	Bedford, Ohio	D, E, F	B, D
Harshaw Chemical Co.	Cleveland	D, E, F	B
McGean Chemical Co.	Cleveland	D, E, F	B
Mallinckrodt Chemical Works	St. Louis	E	A
Nuodex Products Co., Inc.	Elizabeth, N. J.	B, D, F	B, D
The Pyrites Co., Inc.	Wilmington, Del.	A, B, C, D, E	G
Shepherd Chemical Co.	Cincinnati	C, E, F	B, C, G
Standard Oil Co. of Calif.	San Francisco	F	A
Frederick A. Stresen-Reuter, Inc.	Chicago	C, D, E, F	B, D, H
Witco Chemical Co.	Chicago	F	B

[1] Abbreviations: A, metal (rondelles, granules, powder); B, fines; C, oxide; D, hydrate; E, salts; F, driers; G, ore, alloy, or concentrate; H, carbonate.

Co. also has worked out the process for the Blackbird ore. Fredericktown could be counted on for ½ million lb of cobalt annually, once it gets rolling with the new process.

By far the best immediate United States prospect for large amounts of cobalt is the Blackbird mine of the Calera Mining Co., which is operating for Howe Sound Co., in Lemhi County, Idaho. The Blackbird district is in east-central Idaho about 20 miles west-southwest of Salmon. Cobalt was discovered there in about 1901, but there was little interest in the metal until 1917 to 1920. In 1920, the Haynes-Stellite Co. deposit was explored, and 55 tons of concentrates containing 17.74 pct cobalt were produced from 4000 tons of ore. From 1938 to 1941 the Uncle Sam mine was worked, producing about 3657 tons of ore from which copper, gold, and silver was recovered. The U. S. Bureau of Mines and the U. S. Geological Survey mapped and explored the district with diamond drill and bulldozer between 1942 and 1946. In 1943, Howe Sound Co. became interested and took an option on a group of claims. The exploration programs indicated large tonnages of ore and in 1945 Howe Sound began the Calera adit to explore the Chicago zone. Since then plans have gone forward to open the mine, and recovery of the metal from the ore is expected to commence in 1951.

The mineralized rock contains chalcopyrite, cobaltite, pyrite, pyrrhotite, and gold. Research work has been completed on concentrating the ore and recovering the metal. U. S. Bureau of Mines studies disclosed that the high sulphide ores were amenable to treatment by selective flotation, supplemented by low-temperature calcination of the cobalt-iron concentrate and reflotation of the cobaltite. The Howe Sound Co. plans to go ahead with a standard milling procedure, with the exception that flotation is preceded by long conditioning at elevated temperatures. No calcination will be used. This special conditioning serves to sharpen the flotability differential between the chalcopyrite, cobaltite, and pyrite. In succession separate copper, iron, and cobalt concentrates will be made. The iron will go to tailing, the copper will go to the American Smelting & Refining Co. Garfield smelter; and the cobalt will go to a new plant being erected at Garfield. A new process developed by the Chemical Construction Co. for the treatment of the cobaltite concentrate as mentioned in connection with the Fredericktown project will be used to recover the cobalt metal.

Present plans call for production of cobalt metal by September 1951. Capacity will be 2 million lb per year, with the concentrator rated at 600 tons per day. Washington has asked the Howe Sound Co. to increase the production rate to 1000 tons per day, which would mean better than 3 million lb of cobalt per year. It is likely that this will go through although confirmation has not been made.

Surveying the general cobalt picture for the next two years, it is apparent that if the demands of the military for cobalt are not exorbitant, there will be adequate supplies for all essential manufacturing. Furthermore, by 1953 there should be enough production to meet normal consumer demand as well as military requirements. Barring submarine warfare in the Atlantic, 7½ million lb from the Congo can be counted upon. Nearly 900,000 lb a year is being purchased for the stockpile to be delivered by November 1952 from Bou-Azzer, Morocco. The production record of this district and potential reserves indicate that similar tonnages can be counted upon annually. About 3 million lb from Blackbird, ½ million lb from Fredericktown, and ¾ million lb from Cobalt, Ont. can be anticipated. Blackbird production is set at 3 million lb because this is the amount that will be produced pending certain Government guarantees. Negotiations are under way in Washington to procure cobalt from Cobalt, Ont., so that ¾ million lb by 1953 is likely. Table VI shows that 12½ million lb of cobalt would be available annually by 1953. Compared to the 1949 supply of 8 million lb, this is a liberal margin. Table VII shows cobalt refiners or processors in the United States.

The present emergency is stimulating production of cobalt and the development of adequate smelting and refining processes, which will be available to supply industry requirements when and if a normal economy ever returns. There will be many new commercial uses for cobalt following this period of armament, as occurred after World War II.

Table VI—Estimated Annual Available Supplies of Cobalt to the United States by 1953 If Proposed Program is Completed

Source	Cobalt, Lb
Belgian Congo	7,500,000
French Morocco	900,000
Blackbird*	3,000,000
Fredericktown**	500,000
Cobalt, Ont.	750,000
Total	12,650,000

* Calera Mining Co., of Howe Sound Co.
** St. Louis Smelting & Refining Co., of National Lead Co.

Classification of Block Caving And Draw Methods

by F. S. McNicholas

Various methods of block caving and draw practice are classified and the advantages, disadvantages, rock, treatment, size of orebody, costs, and profit comparisons, and other factors which determine the selection of the proper method are outlined.

IN the writer's opinion the term "block caving" has been rather loosely applied to a general principle of mining, and therefore detailed classification of caving methods and draw, together with the characteristics required for each method, will fill a desired need.

Selection of caving method should be made only after a study of all governing conditions, such as size of orebody, lateral and vertical extent of orebody, attitude, treatment demands, possible resultant pressures, and cost and profit comparison of methods.

Successful caving methods can be classified under six main headings: (1) Full-protected block caving, (2) semi-protected block caving, (3) unprotected block caving, (4) full-panel caving, (5) half block-panel caving, (6) block-panel caving. The methods and factors governing choice of method are shown in Table I.

Full-protected Block Caving

Full-protected block caving consists of developing, undercutting, and drawing to exhaustion a block of ore of predetermined size without coordination of mining or draw with adjoining areas. The block is mined and drawn as an independent unit (Fig. 1).

Full-protected block caving demands that mining and draw cannot be done against unconsolidated capping in adjoining caved and exhausted blocks. Two sides of block are against solid. Caved capping must be allowed to consolidate for a suitable period, usually not less than one year. Pillars may be left in lieu of consolidated capping (Fig. 2).

The advantages of full-protected block caving are: (1) A definite area is developed, mined, and drawn as a unit so that all work may be completed, the block drawn to exhaustion in the minimum length of time, and the area then abandoned. (2) Production is not restrained because of coordination with adjoining areas. (3) Records of draw and caved reserve of ore are good. Upon abandonment of the block, accountability for production is closed out with no carry-over to adjoining areas.

The disadvantages of full-protected block caving are: (1) Side dilution from the draw adjacent to the consolidated capping may be serious, particularly where capping does not consolidate sufficiently in the allotted time. (2) Boundary weakening is needed on the solid sides of the block except in weak rocks. (3) Maximum cantilever and compression effect to aid caving and fragmentation may not be secured since the draw should proceed from the solid to the consolidated capping to minimize side dilution. However, undercutting from the loose (caved capping) to the solid with a slight initial draw in the same manner will develop cantilever effect. (4) A period of at least one year, more is usually desirable, must be allowed for caved capping to consolidate before mining the adjoining block, depending upon the characteristics of the capping. (5) Capping may not consolidate sufficiently in the time that can be allowed.

Full-protected block caving is indicated where: (1) The ground is weak and heavy and workings

F. S. McNICHOLAS is Consulting Mining Engineer, Denver.

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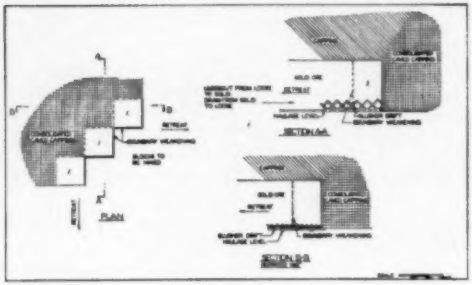
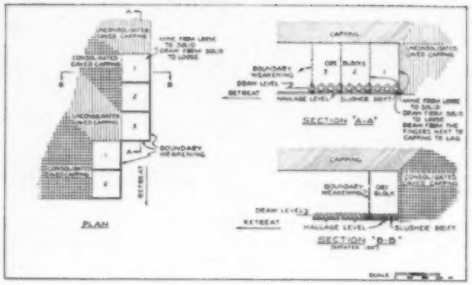




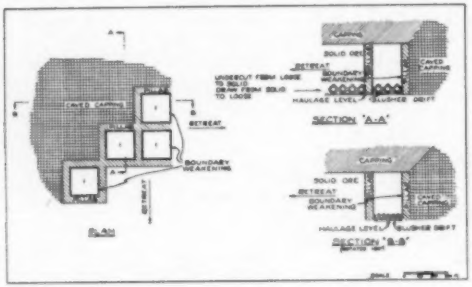


Fig. 1 (upper left)—Full-protected block caving. Fig. 2 (lower left)—Full-protected block caving with pillars. Fig. 3 (right)—Semi-protected block caving.

are difficult and expensive to establish and maintain, and therefore maximum speed of working out a block is necessary. (2) The broken ore consolidates in a short period. (3) The caved capping consolidates firmly in a short period. (4) Different types or grade of ore are desired. (5) Oxidation of caved ore is a serious treatment problem. (6) Ample mining area is available. (7) The capping is waste.

Semi-protected Block Caving

Semi-protected block caving is similar to full-protected block caving except that the block has one side against consolidated capping, one side against newly caved capping, and two sides against the solid, thus allowing the block pattern to retreat along a panel and avoiding to some extent the offset checkerboard pattern of full-protected block caving in one direction. This method is shown in Fig. 3.

The advantages of semi-protected block caving are the same as full-protected block caving plus: (1) A more regular retreat pattern in line is possible. (Retreat is along a panel). (2) The time schedule is more regular as time for consolidation on one side only must be allowed.

The disadvantage of semi-protected block caving is that more side dilution is likely from the side against unconsolidated capping.

Semi-protected block caving is indicated in the following instances: (1) The ground is weak and heavy and workings are difficult and expensive to establish and maintain, and therefore maximum speed of working out a block is necessary. (2) Broken ore consolidates in a short period. (3) Caved capping consolidates firmly in a short period. (4) Oxidation of caved ore may result in a serious treatment problem. (5) Ample mining area is available. (6) The capping is low-grade ore, and the cut-off grade is relatively high.

Unprotected Block Caving

Unprotected block caving is similar to full-protected block caving except that the block has two sides against the solid and two sides against unconsolidated capping (Fig. 4).

The advantages of unprotected block caving are: (1) Extreme flexibility. (2) No time is needed to allow the capping to consolidate. (3) Almost any sequence of mining blocks may be followed that will maintain two sides against the solid.

The disadvantage of unprotected block caving is that side dilution from two sides against unconsolidated capping may be serious.

Unprotected block caving may be indicated where: (1) Maximum speed of developing, mining and drawing an area is desired. (2) Low cost of developing, mining and draw is necessary. (3) Capping is low-grade ore, and the cut-off grade is relatively high (dilution is by low grade of near break-even grade). (4) Low-grade ore mass is to be broken up for a cheap secondary method of recovery such as leaching in place. (5) Cantilever effect is not necessary to cave and fragment the ore. (6) Capping breaks much coarser than the ore, and dilution is thus minimized. (7) Mining area is limited. (8) Broken ore consolidates in a short period.

Full-Panel Caving

Full-panel caving consists of developing and undercutting adjoining blocks of ore in a regular retreating manner with the draw following the mining sequence and regulated to give a retreating draw plane which maintains a toe of broken ore against the block being mined on those sides which have been mined. No block boundary cut-offs or weakening is used. Cut-offs are needed on the periphery of the orebody and to initiate caving.

Almost any sequence of mining adjoining blocks may be followed as dictated by the plan of retreat but should be followed wherever possible in a manner to avoid having block with more than two sides against caved material (Fig. 5).

The advantages of full-panel caving are: (1) The benefit of cantilever and compression action to aid caving and fragmentation may be secured. (2) Boundary cut-offs and weakening are reduced since cut-offs or boundary weakening are needed only along footwall of the orebody (and perhaps to some degree in hanging wall) and to initiate caving action. (3) It is a very flexible method as a predetermined sequence of mining of blocks usually may be altered somewhat without bad effect. However, this

change in sequence may be *local only* as the general established sequence must be followed. (4) There is no time lag awaiting consolidation of caved capping. (5) A very regular sequence of retreat may be established and followed. (6) Minimum cost is secured.

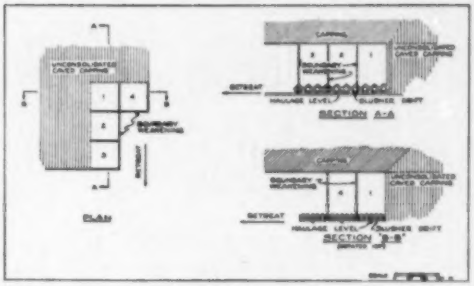
The disadvantages of full-panel caving are: (1) The proper sequence and regulation of the draw is difficult to achieve and maintain and may tie up large potential draw areas reducing production dangerously. Loss of draw points and workings tied up for repair, etc., influence large areas. (2) Since the draw is necessarily from the loose to the solid, good control is essential to prevent funneling through the protecting toe of broken ore into the unconsolidated capping. (3) Large draw areas are required for a given tonnage due to draw coordination with adjoining tonnage and to secure cantilever effect. (4) There is considerable tonnage of broken ore that is unavailable in the protecting toe. Development and equipment for this tonnage is also tied up. (5) Since accountability for remaining broken ore reserve is by the depletion method and cannot be confined to a given area, all of the possible serious errors are thrown into remaining broken ore reserve. These errors may be cumulative, and over a period of time the record of remaining broken ore reserve may be in serious error.

This method should not be used in heavy ground where maximum speed of working out a block is necessary, or where cantilever effect would throw undesirable weight on adjoining areas to be developed and mined. It is applicable where: (1) The ground is medium strong to strong, where maintenance of workings over a considerable period is not a serious consideration, and where maximum cantilever effect is desired, and speed of working out a block is not important. (2) Neither caved capping nor caved ore consolidate to an important degree. (3) Oxidation of caved ore does not present a serious treatment problem. (4) Only one class of ore is to be mined and the grade of ore drawn is limited to the possible variation of draw under the draw control plan. (5) Large mining area is available. (6) Minimum cost is desired.

Half Block-Panel Caving Method

This method, shown in Fig. 6, consists of developing, undercutting, and drawing blocks of ore with two sides against the solid, one side against consolidated caved capping, and the remaining side cushioned from unconsolidated caved capping by a "toe" of broken ore. The retreat may be continuous in one direction and is away from unconsolidated caved capping.

The advantages of half block-panel caving are: (1) The retreat is mainly in one line and is with minimum coordination of draw with any adjoining blocks, so speed of mining the panel could be the maximum if retreat is along the strike. (2) A more regular pattern of retreat is possible. (3) Maximum benefit of cantilever and compression action to promote caving and fragmentation may be secured. Said forces are confined, to some extent, to one line and may be observed and controlled to secure maximum effect. (4) Boundary weakening is needed on one side only in addition to the undercut. (5) Combined top and side dilution should be minimum. (6) There is coordination of draw of fingers in the panel but not with any adjoining panel. (7) There is a minimum of broken ore tied up in the protecting



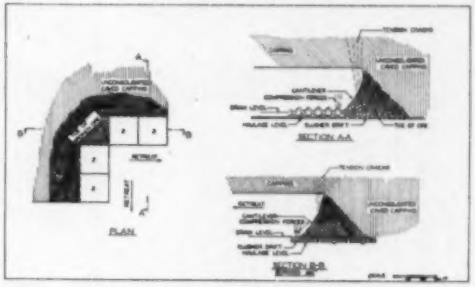


Fig. 4 (top)—Unprotected block caving.

Fig. 5 (bottom)—Full-panel caving.

"toe" on one side only. (8) Accountability superior to any other panel draw.

The disadvantages of half block-panel caving are: (1) If the panel retreat is down dip of a steeply dipping orebody, and the draw is from the loose to the solid, and if there are irregularities in the strength of the orebody, there may be a tendency to form cantilevers up dip and allow capping to draw under ore. If the panel retreat is along strike this does not hold. (2) The method demands consolidated capping on one side to avoid excessive dilution which would result from unconsolidated capping. (3) Mining area may be limited since the succeeding panel must await consolidation of capping.

Half block-panel caving is indicated where: (1) The ground is good and maintenance over a medium term is not a serious problem. It may be used where maximum speed of mining an area is desired. (2) Ore is of good grade and capping is waste. (3) *Controlled* cantilever and compression effect is desired with minimum arching effect. (4) Natural character of orebody favors retreat and cantilever effect is one line, or direction. (5) Retreat is along the strike of a steeply dipping orebody. (6) Caved capping consolidates in a reasonable period of time.

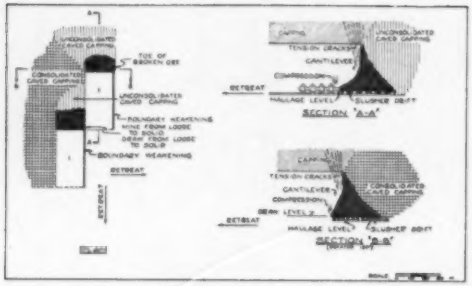
Block-Panel Caving Method

This method consists of developing and mining adjacent blocks of ore as units in a retreating manner with the main draw of the block from solid to loose and regulated to maintain a relationship with adjoining blocks which are being drawn. Cut-offs weaken the boundary on the two sides next to the solid. Undercutting proceeds from the loose to the solid and a slight draw follows the undercutting to establish cantilever effect. After undercutting is completed, the main draw is from solid to loose

Table I. Block Caving Methods and Factors Governing Choice of Method

Method	Rock Characteristics			Working Installation and Maintenance	Speed of Exhaustion of Block	Mining Area and Method	Treatment Demands	Dilution	Recovery	Draw	Accountability	Costs
	Ore	Capping	Footwall									
	1	2	3	4	5	6	7	8	9	10	11	12
Full-protected Block Caving	Weak Heavy Breaks fine Contains binder, damp. Consolidates in short period. Cantilever effect not necessary for caving and fragmentation.	Is waste Contains binder. Consolidates in short period. Caved capping adjoining blocks to be worked allowed to consolidate.	Weak Heavy Difficult and expensive to install and maintain workings.	Difficult and expensive to install and maintain due to 1 and 3.	Rapid exhaustion of block essential due to 1, 3, 4 and 7.	Method is indicated where mining area is not large. Undercutting from loose to solid. Two sides have boundary weakening.	Treatment demands different types of ore or a combination of grade. Oxidation of caved ore kept to minimum.	Dilution kept to reasonable amount.	Good to excellent with proper draw control.	No coordination of draw with adjoining blocks making rapid draw possible. Draw is from solid to loose.	Accountability of metal content the best. No cumulative errors, when block is completed.	Reasonably low costs but higher than full panel. Is a low cost mining method.
Semi-Protected Block Caving	Ore is weak Heavy Breaks fine Contains binder, damp. Consolidates in short period. Cantilever effect not necessary for caving and fragmentation.	Capping is low grade ore so dilution is with low grade not waste. Contains binder. Consolidates in a short period.	Weak Heavy Difficult and expensive to install and maintain workings and equipment.	Difficult and expensive to install and maintain workings and equipment due to 1 and 3.	High speed of exhaustion of block essential due to 1, 3, 4 and 7.	Method indicated where mining area is not large. Undercutting from loose to solid. Two sides have boundary weakening.	Treatment demands different types of ore or a combination of grade oxidation of caved ore kept to minimum.	Dilution from side against unconsolidated capping is greater than with a full-protected block. Dilution may be held to a reasonable amount by strict and proper draw control.	Fair to good with proper draw control.	No coordination of draw with adjoining blocks making rapid draw possible. Draw is from solid to loose.	Accountability of metal content is good. No cumulative errors when block is completed.	Is a low cost mining method. Reasonably low costs about same as full-protected block method.
Unprotected Block Caving	Ore is weak Heavy Breaks fine Contains binder, damp. Consolidates in short period. Cantilever effect not necessary for caving and fragmentation.	Capping is low grade ore of near breakeven grade. Is strong and breaks big and blocky so dilution may be low. Caved capping of adjoining blocks not given time to consolidate.	Weak Heavy	Speed of getting into and out of a block may make installation and maintenance of workings less costly than for full or semi-protected blocks.	Maximum speed of developing and exhaustion of block and area is secured.	Mining area may be relatively small. Undercutting from loose. Boundary weakening on two sides.	Low-grade mass is broken for a cheap secondary method of recovery such as leaching in place. Treatment demands different types of ore or combination of grades. Oxidation of broken ore a minimum.	Dilution may be high or if low recovery is acceptable, dilution may be reasonable.	Recovery will be relatively poor or high dilution must be accepted.	No coordination of draw with adjoining blocks. Rapid draw is possible. Draw is from solid to loose.	Accountability will be poor, but without cumulative errors.	Is very low cost method applicable to low-grade ore.

Full-Panel Caving	Ore is medium strong to strong. Maximum cantilever effect desired to cave and fragment ore. Caved ore contains little binder and does not consolidate to an important degree.	Caved capping contains little binder and does not consolidate to an important degree.	Strong to medium strong.	Cost of maintenance over a long period is not a problem due to 1 and 3.	Speed of exhaustion of block not important due to 1, 3 and 10.	Large area required due to 10. Undercutting from loose to solid. No boundary weakening. Cut off on perimeter of ore body only.	Only one class of ore can be produced. Grade of ore drawn is limited to possible variations of draw as dictated by the draw control plan. Method not suitable if oxidation of broken ore is a problem.	Dilution is a reasonable amount if proper draw controls are maintained. May be less than by other methods.	Recovery is good to excellent with proper draw control.	The full panel draw demands co-ordination with adjoining draw points. Draw is from loose to solid in retreating manner. Maintaining a retreating toe of ore under caved capping.	Accountability of metal content is very poor. The depletion method demanded to establish remaining broken ore reserve throws all errors into remaining broken reserves which over a period may result in drastic error.	Under proper conditions minimum costs will be secured.
Half Block-Panel Caving	Ore is medium strong to strong. Caved ore contains little binder and does not consolidate to an important degree. Maximum controlled cantilever effect can be secured.	Caved capping breaks big or contains binder and will consolidate.	Strong to medium strong.	Maintenance of workings is for a shorter period than for full panel caving.	Speed of exhaustion of block may be relatively rapid.	Mining area may be less than full panel and more than full protected block. Undercutting is from loose to solid. Boundary weakening on one side only.	Only one class of ore can be produced under ordinary circumstances. Limitations of possible grade are less rigid than full panel.	Dilution is a reasonable amount with proper draw controls.	Recovery will be good with proper draw controls.	The draw is half panel. Coordination of draw confined to fingers in the panel. Draw is from loose to solid retreating in one direction only. Protective toe of ore on one side of block.	Accountability of metal content is reasonably good. Accountability comes to a check upon completion of panel.	Low costs will be secured. Less than full protected block. More than full panel.
Block-Panel Caving Is a combination of unprotected block caving and a full panel cave and draw. Is designed to limit stresses to the block and limit the area necessary for full panel draw.	Ore may be medium strong to strong or weak. May or may not contain binder. May or may not consolidate in short period. Cantilever effect can be secured to some extent.	Capping may have almost any characteristics, preferable breaks big with little fines.	Footwall may have any characteristics that apply to caving methods. Preferably strong.	Maintenance of working and installation is for a relatively short period.	Speed of exhaustion of block is rapid. Speed of production from an area is maximum.	Maximum production secured from given area. Weight and stresses confined to the block. Undercut from loose to solid.	Only one class of ore can be produced. Oxidation of broken ore is kept to a minimum.	Dilution is reasonable to high. Proper rigid draw controls a must. After undercutting is complete, draw is from solid to loose with fingers against loose lagging 25 to 30 pct.	Recovery good to medium. Proper rigid draw controls a must.	Slight draw following the undercutting from loose to solid allows cantilever effect to develop. Main draw after undercutting completed is from solid to loose. This is essential.	Accountability of metal content is poor but better than full panel.	Costs will be more than full panel, about the same as unprotected block.



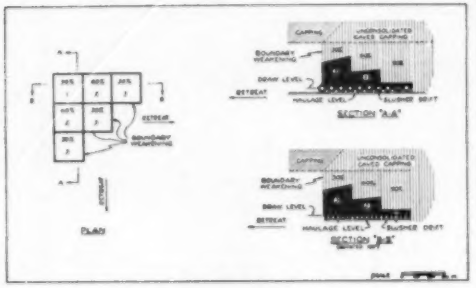


Fig. 6 (top)—Half block-panel caving.

Fig. 7 (bottom)—Block-panel caving.

with the fingers next to the loose lagging about 25 pct behind those next to the solid. When about 25 to 40 pct of tonnage is drawn, the entire ore in the block has caved and is about the same "fluidity" as the unconsolidated capping so further draw should be vertical.

The sequence of mining of blocks would be on an offset checkerboard pattern to give maximum production. This method is shown in Fig. 7.

The advantages of block-panel caving are: (1) Allow draw to retreat from the solid toward the broken ore cushion. With proper control of the draw in this manner, by the time the draw has progressed to the broken ore cushion, the ore in the entire block has caved and the density (fluidity) of the entire mass is about the same, so tendency of the draw would be to pull vertically. (2) Dilution and loss of ore should be less than most other methods excepting half block-panel. (3) Allows draw without time lag needed for consolidation of adjoining caved capping. (4) High continuous rate of production.

The disadvantages of block-panel caving are: (1) Full cantilever effect is not secured. (2) It requires weakening on two sides. (3) Draw is essentially a panel draw and if followed strictly will result in tying up large areas.

This method is indicated where: (1) The ground is strong to medium strong and maintenance of workings over a medium term is not a serious problem. (2) Maximum speed of exhausting an area is not essential. (3) Capping is strong and breaks bigger than ore with little fines. (4) Capping does not consolidate to a marked degree in a reasonable time. (5) Mining area is limited and maximum production must be secured from a given area. (6) Broken ore does not consolidate in a short period. (7) Draw must be from solid to loose to minimize dilution from adjoining unconsolidated caved capping or sand fill or other filled stopes.

Determination of Characteristics

Rock Characteristics: The characteristics of the rock of the orebody, capping and footwall may be determined by: (1) Detailed examination and classification of diamond drill core in respect to fracture pattern, fracture cleavage, jointing, texture, oxidation, mineralization, binder, etc. (2) Comparison of diamond drill core with core from orebodies of known characteristics by direct comparison of colored transparencies. (3) Detailed examination and classification of characteristics of excavation, and perhaps some trial blocks, compared with characteristics of excavation in known orebodies having known characteristics. (4) Comparison of colored transparencies of excavations comparing known characteristics with those under study. (5) Detailed study of caved ore and caved capping to determine size of fragments, amount of fines, amount of binder, amount of moisture, amount of consolidation over a period of time. (6) Application of observed general characteristics of known orebodies compared to the orebody under study.

Treatment Demands: May be determined by test work concerning the metallurgy involved, the desirable ratios of different ores or grades and the marketing demands.

Oxidation of caved ore may be a determining factor in arriving at the method to be used.

Mining Area and Speed of Production: Tonnage demands and speed of production from a block or an area may indicate the desirable method.

Draw: Experimental work on draw in scale models approaching actual proven conditions as closely as possible may give useful information, however, such work should have some background of practical operation to be of value.

Experimental work of others should be reviewed for possible application.

Experimental work will generally indicate a lower dilution than will be secured in practice.

Dilution and Recovery: It is difficult and perhaps impossible to predetermine these factors to an absolute degree. They may, however, be determined to a relative degree, and in such a manner be useful in determining method.

Accountability: Accountability of metal content probably will be of minor importance in determining method, however, small concessions may be made to secure good accountability.

Costs: All methods give reasonably low costs. Determined characteristics will largely determine method with its accompanying costs.

Acknowledgment

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Radiotracer Studies on the Interaction of Dithiophosphate with Galena
The Depressant Action of Phosphate

by G. L. Simard, M. Burke, and D. J. Salley

The depression of dithiophosphate by phosphate was investigated by correlating radiotagging measurements of competition in sorption with measurements of flotation. Competition in sorption was as complex as the sorption of the individual ions, thus strengthening the conclusion that more than one process was involved in the interaction of the ions with galena. The depression of flotation was chiefly a result of reducing the amount of dithiophosphate sorbed.

A PREVIOUS paper[1] described experiments on the interaction of dithiophosphate with galena and suggested that the process had characteristics of both chemisorption and surface chemical reaction. In order to add to this knowledge, a study has been made of the competition of a second ion with the dithiophosphate for a galena surface. Use again was made of radiotracer methods. Since radiophosphate was readily available, it was chosen as the competing ion. It is a feature of the tracer method that, in spite of similarity in the chemical behavior of the ions, the measurement of one ion in the presence of another can be made as simply as if only one ion species were present.

In order to study the competition effectively, it naturally was necessary to examine the interaction of phosphate with galena alone as well as in the presence of dithiophosphate; consequently, this report consists of two parts. The first is devoted to a study of the rates, the isotherms, the desorption, and the exchange in the phosphate-galena system in the same manner as was done for the dithiophosphate-galena system. The second portion is concerned with the studies on the competition. Flotation experiments on the three-component system are also outlined and the relationships involved are discussed briefly.

The experimental method was that of the previous paper. The preparation of the galena, the radioactive and the nonradioactive dithiophosphate, and the procedures for radioactive counting, for sorption, and for flotation measurements were identical to those previously employed. The radiophosphate* was used in the form of disodium phosphate solution.

In the competition experiments the effect of the phosphate ion in depressing dithiophosphate uptake was followed by holding the initial concentration of nonradioactive phosphate constant while varying the initial concentration of the radioactive dithiophosphate. Counterfashion, the depression of phosphate by dithiophosphate was obtained by keeping the initial nonradioactive dithiophosphate ion concentration constant and altering the initial concentration of the radioactive phosphate ion.

All solutions were buffered at a pH of 7.8 ± 0.1 by 20×10^{-5} mol per cc of sodium carbonate. Measurements with radiocarbonate indicated that little or no carbonate could be sorbed in the presence of

G. L. SIMARD, M. BURKE, and D. J. SALLEY are associated with the Research Div., American Cyanamid Co., Stamford, Conn.

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* For early experiments (1943-1944), the elementary red phosphorus and the phosphate were obtained from the cyclotron group at the Crocker Radiation Laboratory of the University of California, Berkeley, through the courtesy of Dr. Joseph W. Hamilton. For more recent work (1946-1949), both radiophosphate and phosphorous have been supplied by the Oak Ridge National Laboratories on allocation from the U. S. Atomic Energy Commission.

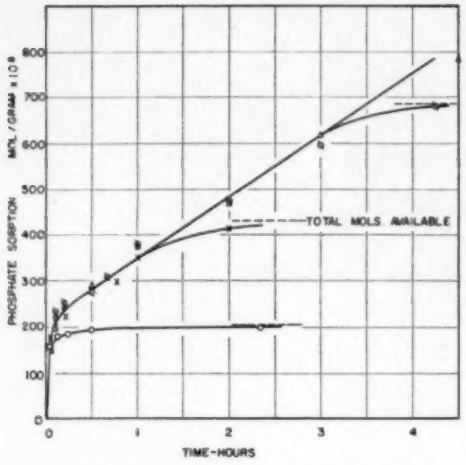

Fig. 1—Rate of phosphate sorption on unleached galena No. 3.

Initial phosphate concentrations: ○, 8.1×10^{-6}; X, 17×10^{-6}; ∅, 27×10^{-6}; △, 50×10^{-6} mol per cc.

phosphate, although in its absence some sorption was possible.

Interaction of Phosphate and Galena

Rates: Figs. 1 and 2 show the essential facts concerning the rates of uptake of phosphate by galena. A rapid initial sorption occurred which was essentially complete in 10 to 12 min. In nitrogen no following secondary process took place on leached galena. However, in air there did occur a slower, secondary uptake which proceeded at a constant and identical rate on both unleached and leached galena. The rate of this secondary interaction was independent of concentration until the phosphate in solution was relatively low. After very long periods of time, the phosphate concentration asymptotically approached very low values, as is indicated in Table I. (The sorption in nitrogen was the same whether or not leaching had been done in the presence or absence of air).

Isotherms: The dependence on concentration of the amount of phosphate sorbed in the rapid initial process at the end of 12 min is shown by the typical results of Fig. 3 for ground mineral and for single crystals, respectively. In both cases sorption took place at the lowest concentration studied, 0.2×10^{-6} mol per cc.

Independence of concentration was generally attained at a low concentration of phosphate, and the low concentration region was not defined sufficiently well to permit fitting to a Freundlich type equation; this was in contrast to the case of the dithiophosphate isotherms. However, in similarity with the dithiophosphate observations, a considerable part of the sorption on unleached galena was by leachable material. Also, the scatter of the data for single crystals was again correlated with the appearance of stain on the crystal face.

There was a marked increase in sorptivity on exposure of the mineral to air. For example, on galena No. 5, 55×10^{-6} mol per g more phosphate was sorbed by leached mineral when it was dried in air subsequent to leaching than when the mineral was kept wet and sorption measurement made immediately. The amounts of phosphate corresponding to the horizontal portions of the 12-min isotherms for various unleached galena preparations are given in Table II and illustrate the variability between similar preparations of unleached mineral.

Desorption: Desorption measurements for galena No. 2 in water and galena No. 3 in 20×10^{-6} mol per cc Na_2CO_3 are given in Table III. The sorption times were 12 min for No. 2, and 26, 36, and 182 min for No. 3. The data for preparation No. 3 are average values from five measurements.

The desorption was unaffected by either the quantity of phosphate initially on the galena or the time used for its sorption. The desorption of phosphate from ground, unleached galena took place at a slower rate and to a lesser extent than the desorption of dithiophosphate.

Table I. Phosphate Concentration After Long Periods of Time

Galena No. 3

(20×10^{-6} mol per cc Na_2CO_3)

Initial Concentration Mol per cc x 10^6	Adsorption Time, Hr	Final Concentration Mol per cc x 10^6
8.1	4	0.14
19.0	17	0.07
25.0	23	0.03
27.4	42	0.02

Table II. Phosphate Sorption by Unleached Galena at 12 Min

Galena	Phosphate Sorbed Mol per g x 10^6
No. 2	240
No. 3	180
No. 4	850
No. 5 (unleached)	240
No. 5 (leached)	85
Single crystals	(50)[a]

[a] For 1000 cm^2 per g on basis of 0.05×10^{-6} mol per cm^2 at a concentration 20×10^{-6} mol per cc.

Table III. Desorption Measurements of Phosphate Sorbed on Galena

Galena	Phosphate Initially Sorbed Mol per g x 10^6	Desorption Time	Mols Desorbed Mol per g x 10^6	Concentration of Desorp. Soln. Mol per cc x 10^6
2	363	14 min	0.53	0.02
	363	25 min	0.40	0.02
3	176; 604	1 hr	0.25	0.01
	176; 604	5 hr	0.75	0.03
	176; 604	13 hr	1.80	0.06

Table IV. Exchange of Phosphate Between Galena and Solution, 1 Min Exchange Time

(20×10^{-6} mol per cc Na_2CO_3)

Sorption Time, Min	Total Mols x 10^6		Soln. Concentration Mol per cc x 10^6	Fraction Exchanged
	In Soln.	On Galena		
5	356	179	17	0.11
13	328	215	16	0.12
80	220	349	8.6	0.17
120	146	442	5.0	0.27

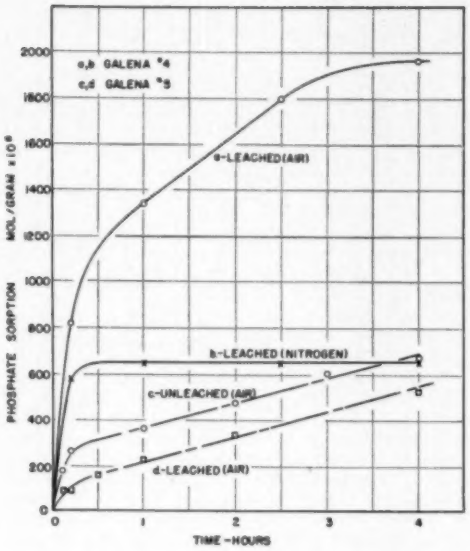

Fig. 2—Effect of air and of leaching on the rate of phosphate sorption by galena.

Curves a, b: Effect of air and of N_2 on sorption. Initial phosphate concentrations: a, 100×10^{-6}; b, 50×10^{-6} mol per cc.

Curves c, d: Effect of leaching on sorption in air. Initial phosphate concentrations: c, 50×10^{-6}; d, 40×10^{-6} mol per cc.

For comparison, the solubility of $Pb_3(PO_4)_2$ was measured in water and 20×10^{-6} mol per cc Na_2CO_3. In both media, the equilibrium phosphate concentration was $0.097 \pm 0.002 \times 10^{-6}$ mol per cc.

Exchange: The extent of exchange in one minute between sorbed radioactive PO_4 on ground galena and inactive PO_4 in solution was studied. Any sorption which might have ocurred during the exchange measurement was negligible. The data as a function of time of sorption and the mols of phosphate in the solution and on the mineral are shown in Table IV.

For purposes of comparison, measurements were made of the exchange of PO_4 between solution and freshly precipitated $Pb_3(PO_4)_2$, Table V. These were necessarily at higher actual mols in precipitate.

Similar to the dithiophosphate case, exchange occurred here between phosphate ion and the freshly precipitated lead phosphate as well as with the sorbed phase on galena. Comparisons are again handicapped by the difficulties of attaining comparable conditions, so that definite conclusions cannot be drawn.

In the previous investigation on the interaction of dithiophosphate and galena, it was found that the heterogeneous, complex nature of the galena surface played a determining role in the rapid initial process. Because of this, it was natural that the rapid initial sorption of phosphate should possess points of similarity with that of dithiophosphate. Characteristic of both interactions were the following: completion in about 10 to 12 min, variability in the amounts taken up by the different galena preparations, decreased sorption on leaching, increased sorption on weathering, finite exchange rates, and slow desorption to produce low solution concentrations. From these similarities, it was expected that the phosphate-galena interaction should, as in the case of dithiophosphate, show characteristics of both chemisorption and chemical reaction.

Table V. Exchange of Phosphate Ion with Lead Orthophosphate

Soln. Concentration Mol per cc x 10^6	Total Mols Present x 10^6		Exchange Time Min	Fraction Exchanged
	In Soln.	In Precip.		
16.2	404	200	10	0.25
8.2	204	200	10	0.24
4.1	102	204	10	0.16
1.6	41	204	10	0.07

In line with this, it was possible to dismiss physical adsorption because of the finite exchange rate, and because of the failure, on desorption, to approach the same equilibrium concentrations existing during sorption. However, neither chemical reaction nor chemisorption could be excluded. On the one hand, chemisorption was plausible because on leached galena, for example, the amount taken up (80×10^{-8} mol per g) was approximately that calculated (90×10^{-8} mol per g) for close packing of PO_4 groups (18Å² from Fischer-Hirschfelder models) in a monolayer on a surface of 1000 cm² per g. This was also approached in the uptake by single crystals, namely an observed value of 0.06×10^{-8} mol per cm² as compared to the calculated of 0.09×10^{-8} mol per cm². On the other hand, chemical reaction leading to compound formation was indicated because the amounts taken up on unleached galena were equivalent to several monolayers, and furthermore, the concentration reached on desorption (0.06×10^{-6} mol per cc) compared favorably with that expected from the solubility of lead orthophosphate. Evidently, therefore, the initial interaction with phos-

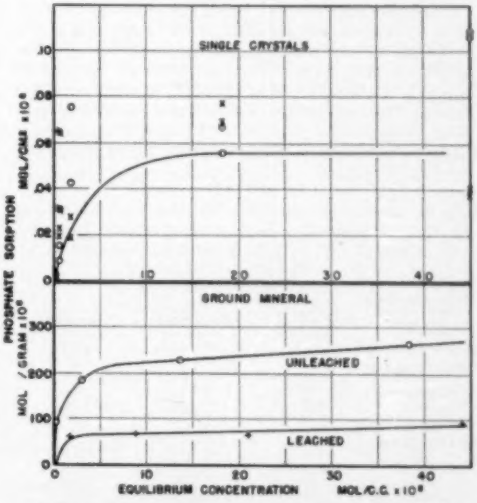

Fig. 3—Sorption isotherms (25°C) of phosphate on single crystals and on ground galena No. 5. Sorption time, 12 min.

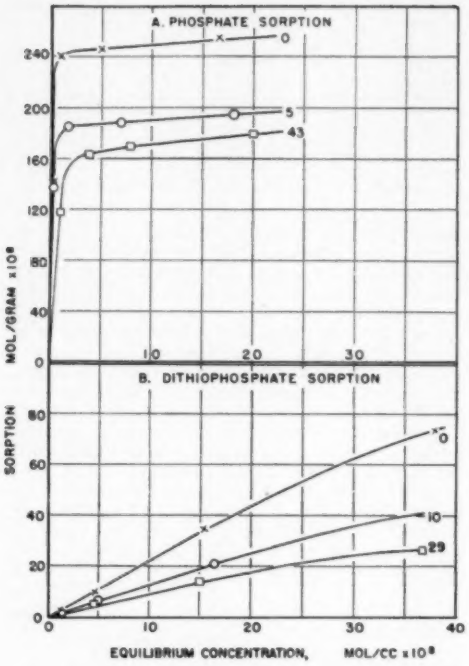

Fig. 4—Competition isotherms (25°C). Unleached galena No. 2; sorption time 12 min.

A. Phosphate sorption in the presence of dithiophosphate; initial dithiophosphate concentrations of 0, 5, and 43x10⁻⁸ mol per cc.

B. Dithiophosphate sorption in the presence of phosphate; initial phosphate concentration of 0, 10, and 29x10⁻⁸ mol per cc.

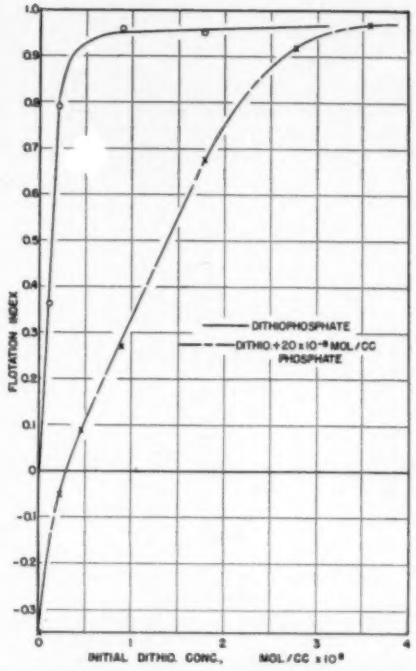

Fig. 5—Flotation of galena No. 2 by dithiophosphate, alone and in the presence of phosphate.

phate showed the same complexity observed with dithiophosphate.

Of considerable interest was the observation that no fixed ratio of phosphate to dithiophosphate could be found for the amounts taken up on the various galena preparations. Comparison of phosphate sorption, Table II, with that of dithiophosphate, Table II,[1] shows that the phosphate to dithiophosphate ratio varied from 1.0 to 0.33. This strongly suggests that if the interaction was a single process in one case, more than one process was involved in the other.

A feature of the phosphate interaction in air was the long, slow sorption following the rapid initial process. This was characterized by (1) dependence on dissolved oxygen in solution, (2) independence of rate on phosphate concentration (except at relatively low concentrations), and (3) equality of rates on leached and unleached mineral. These facts show that the slow process resulted from a primary reaction of oxygen in solution with lead sulphide to form complexes or compounds, which then reacted with phosphate. The phosphate eventually reached low concentrations (0.1 to 0.02x10⁻⁸ mol per cc) because the oxidation made lead continuously available. Since these final concentrations were comparable to the solubility of lead orthophosphate, it therefore appeared that the slow process resulted eventually in the formation of lead orthophosphate. The desorption also produced values of the same order of magnitude. Being the least soluble of the phosphates, the orthophosphate would be the eventual end product of the slow reaction.

Table VI. Comparative Depression of Phosphate on Unleached and Leached Galena No. 5 by Dithiophosphate

Dithiophosphate Initial Conc. Mol per cc x 10⁸	Phosphate* Equil. Conc. Mol per cc x 10⁸	Phosphate* Sorption Mol per g x 10⁸	Depression, Pct
Unleached Galena			
0	29.5	210	0
22.5	30.6	188	10
73.0	32.9	142	32
130.	38.9	22	89
Leached Galena			
0	33.9	105	0
22.0	35.6	70	33
73.2	37.7	27	74
146.	38.6	9	91

* Initial Concentration Phosphate, 40x10⁻⁸ Mol per cc (20x10⁻⁸ Mol per cc Na_2CO_3)

Depression of Dithiophosphate by Phosphate

Sorption: The effect of phosphate on dithiophosphate sorption was studied by measurements of the 12-min sorption isotherms of each ion in the presence of the other. Such measurements could have been invalidated by exchange in solution of the tagged atom between the two ion species; actual determinations, however, proved that such exchange did not occur in the period of one hour over the concentration range employed.

Competition took place for an unleached galena surface as shown by the typical sorption data for galena No. 2 in Fig. 4 and for galena No. 5 in Tables VI and VII. Phosphate was a relatively stronger depressant than dithiophosphate. A point of difference existed between galenas No. 2 and No. 5 in that on No. 5 complete depression of either ion by the other was possible, while on No. 2 only partial depression could be obtained. This is further evidence of the variability and heterogeneity of the mineral surface. Evidently, on galena No. 2 a surface available to phosphate alone and a surface available to dithiophosphate alone must have been present in addition to a surface of competition.

The effect of leaching the mineral before sorption is illustrated in Tables VI and VII. After leaching, dithiophosphate was not as strongly depressed by phosphate and, conversely, phosphate was more strongly depressed by dithiophosphate. The degree of competition varied, therefore, with the composition of the mineral surface.

Flotation: In the presence of a relatively large concentration of phosphate, the flotation of unleached mineral was strongly depressed as illustrated by data on galena No. 2 in Fig. 5. The depression of the blank value, or weight floated in absence of collector, indicated that sorbed phosphate may have rendered the surface more wettable. Flotation increased with increasing collector concentration and asymptotically approached the maximum value at a concentration about eight times that when no phosphate was present.

Fig. 6 shows the effectiveness of small concentrations of phosphate in depressing flotation when the amount of collector sorbed was the minimum necessary for complete flotation. Measurements were made with a dithiophosphate concentration of 0.45×10^{-6} mol per cc corresponding to the knee of the normal flotation curve. On adding small quantities of phosphate, flotation was sharply depressed; reduction to one half the original value occurred at a phosphate equilibrium concentration of only 0.15×10^{-6} mol per cc.

Leached and unleached galena No. 5 showed a definite differences in flotation. For example, in a solution containing 2.5×10^{-6} mol per cc of dithiophosphate, the flotation of unleached galena was decreased 40 pct by the addition of 30×10^{-6} mol per cc phosphate, while that of leached galena appeared to be unaffected.

Discussion

The data on the competition of phosphate and dithiophosphate for a galena surface showed that the competition process was equally as complex as the interaction of the substances individually. In view of this, it was not surprising that the data were not amenable to quantitative treatment. By proper interpolation and arithmetical transformation, it was possible to calculate the equilibrium concentrations of the respective ions in solution for any given amount of dithiophosphate and of phosphate on the galena sample. These are the quantities necessary for any type of correlative calculation. The data of the interactions individually did not allow prediction of the relative amounts of each sorbed when in the presence of one another. Certainly, no prediction of the relative sorption from one galena preparation to another was possible. In no case could the entire competition picture be represented by isotherms

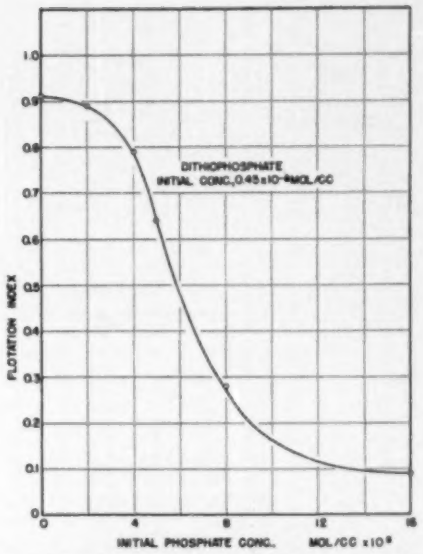

Fig. 6—Depression of flotation of galena No. 2 by phosphate. The initial dithiophosphate concentration was near the minimum necessary for complete flotation in the absence of phosphate.

derived on the assumption that two components were competing for one or for two surfaces. Finally, no satisfactory treatment of the data was obtained by attempts to apply solubility product considerations assuming coprecipitation of the two substances as lead salts. These points generally strengthen the prior conclusion that more than one process is involved in the interaction of these agents with galena. Also, they suggest that the quantitative aspects of competition in systems of this type are only learned by direct measurement. The value of radiotracer or other methods which directly provide quantitative knowledge of the amounts taken up and the resulting equilibrium concentrations cannot be over-emphasized, especially in view of the importance of competition in activator as well as depressant action.

Flotation of a mineral occurs when the amount of collector sorbed is sufficient, by virtue of its hydro-

Table VII. Comparative Depression of Dithiophosphate on Unleached and Leached Galena No. 5 by Phosphate

Phosphate Initial Conc. Mol per cc x 10^6	Dithiophosphate [a] Equil. Conc. Mol per cc x 10^6	Dithiophosphate [a] Sorption Mol per g x 10^7	Depression, Pct
Unleached Galena			
0	59.9	830	0
18.8	48.2	344	35
37.5	59.2	64	88
75.0	61.0	28	95
Leached Galena			
0	49.6	231	0
18.8	54.1	139	40
36.6	57.7	65	72
73.2	59.0	38	84

[a] Initial Dithiophosphate Concentration, 62.4×10^{-6} Mol per cc (20×10^{-6} Mol per cc Na_2CO_3)

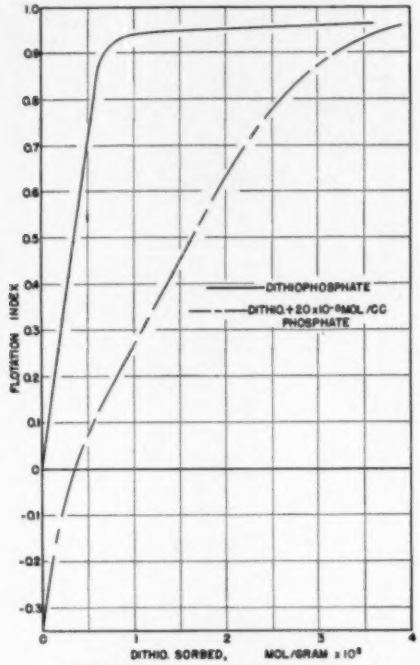

Fig. 7—Relation of the amount of dithiophosphate sorbed to the flotation for galena No. 2.

phobic groups, to maintain attachment of the mineral particles to the air-water interface. Flotation in solutions containing dithiophosphate and phosphate should accordingly be determined, not by the concentration of the dithiophosphate, but rather by the amount of dithiophosphate taken up, provided that the sorbed phosphate has no hydrophilic influence. A plot of flotation index versus dithiophosphate on the galena in the absence and in the presence of phosphate is shown in Fig. 7, as constructed from sorption and flotation data. It is clear that the amount of dithiophosphate sorbed is not solely controlling, for if such were the case the two curves would coincide. Evidently, the hydrophilic character of the sorbed phosphate has a significant influence on the effectiveness of a given sorbed quantity of dithiophosphate. This indicates that the characteristics of the depressing ion must be considered as well as the amount of collector taken up. Fig. 7 is also of interest as an example of the information which can be obtained by simple tracer measurements. As previously pointed out, complete flotation occurred with a relatively small amount of dithiophosphate on the surface when no phosphate was present (monolayer formation was estimated at 45×10^{-8} mol per g of collector). The strong depression by phosphate at concentrations corresponding to the knee of the dithiophosphate curve was obviously the result of the critical dependence of flotation on the amount of collector sorbed in this region. The mineral surface was also a determining factor, since leaching increased the relative amount of dithiophosphate to phosphate taken up and correspondingly increased flotation.

All of the considerations of this and the first paper on the interaction of dithiophosphate with galena have been mainly focused on the initial rapid process, since it appears of primary interest in flotation. The complex nature of the interaction has been indicated. For additional clarification, investigation at the lowest possible concentrations and surface coverages would be valuable since only small amounts are necessary to produce maximum flotation. In addition, measurements at several temperatures should prove helpful because they would permit estimates of the activation energies and the heats of the processes involved.

Summary

A previous study[1] of the interaction of dithiophosphate with galena was extended by a study of the depressant action of phosphate on this system. The depression was investigated by correlative measurements of the competition of each ion for galena in sorption and in flotation. Using selective radiotagging methods, measurements of one ion in the presence of the other could be made as simply as if only one species had been present. It was necessary for the interpretation of these data to study also the interaction of phosphate alone with galena.

The interaction of phosphate with galena possessed considerable similarity to that of dithiophosphate. Characteristics of both chemisorption and surface chemical reaction were evident. The complexity of the interaction resulted largely from the heterogeneous nature of the galena surface.

Competition in sorption between dithiophosphate and phosphate was equally as complex as the sorption process of the individual ions. Although qualitatively the results were those expected from the observed behavior of the separate ions, no satisfactory quantitative treatment of the data could be made. This strengthened the conclusion that more than one process was involved in the interaction of these agents with galena, and suggested that the quantitative aspects of such systems may only be determined by direct measurements.

The flotation data generally confirmed the hypothesis that the depressant action of phosphate was a result of reducing the amount of dithiophosphate taken up by the mineral. The hydrophilic character of the sorbed phosphate, however, appeared to influence the effectiveness of the sorbed collector for flotation. Leaching of the mineral reduced the effectiveness of the depressant by increasing the relative amount of collector sorbed, thus emphasizing again the dominant role of the mineral surface in flotation processes.

Acknowledgment

We wish to express our appreciation for encouragement and support from S. J. Swainson and R. B. Barnes, for helpful discussion with R. B. Booth, S. A. Falconer, J. E. Carpenter, and for technical assistance from various other members of these Laboratories during this investigation.

Reference

[1] G. L. Simard, J. Chupak, and D. J. Salley: Radiotracer Studies on the Interaction of Dithiophosphate with Galena. *Transactions* AIME (1950) **187**, 359-364; *Mining Technology* (March 1950) TP 2815.

Geology, Mining, And Uses Of Strategic Pegmatites

by Richard H. Jahns

Fig. 1—Principal pegmatite areas and districts of the United States.

Such minerals as beryl, lepidolite, sheet muscovite, spodumene, and tantalite-columbite are obtained chiefly from pegmatite bodies that are internally zoned. As shown by examples of such pegmatites from the Southwestern and Southeastern States during World War II, studies of their internal and external structure have value in prospecting and the planning of exploration, development, and mining.

GRANITIC pegmatite deposits are the chief source of commercial feldspar, sheet mica, beryllium, tantalum-columbium, and lithium minerals, and certain types of kaolin. They also have yielded significant quantities of cassiterite, gems, scrap mica, molybdenite, tungsten minerals, uranium-thorium and rare-earth minerals, and zircon, either directly or as the sources of placer deposits. The output from pegmatite mines in the United States is small as compared to other mineral products in terms of bulk or value, and much of it comprises minor metals and nonmetals. Nevertheless, pegmatite minerals play a vital part in domestic industrial economy, particularly in the ceramic and electrical industries. Numerous special purpose uses also are important, even though they require small quantities of raw materials.

Never was United States dependence upon pegmatite mining more clearly emphasized than during World War II, when greatly expanded demands and uncertainties of foreign sources of supply caused much concern. Such commodities as beryl, tantalite, and sheet muscovite remained high on the critical list for months at a time, and for some minerals there was a real struggle to achieve and maintain a favorable ratio of supply to demand. Satisfactory stock piles of certain sizes and qualities of sheet mica, for example, were not built up until late in 1944, when wartime requirements already had begun to level off or even to decrease. Production of all the strategic pegmatite minerals from domestic deposits during World War II constituted only 10 pct or less of the total domestic consumption of these minerals, but the importance of this contribution should not be minimized. Domestic production often represented the difference between increase and further reduction of existing stocks of commodities already in seriously short supply, and in the early stages of the war it was the only assured source if imports were cut off.

During the recent war period, the U. S. Geological Survey carried on a program of pegmatite studies in the New England and Southeastern States, in the Black Hills area of South Dakota, in the Rocky Mountain region of Wyoming, Colorado, and New Mexico, and in various parts of Washington, Idaho, Utah, Nevada, Arizona, and California as shown in Fig. 1. Several papers and reports already have appeared as an outgrowth of this work,[1-10] and others are in preparation. This paper outlines some results of data obtained by the writer during the period 1942 to 1946, when he was engaged in studies of pegmatites in the Southeastern, Southwestern, and Rocky Mountain States.

Beryllium Minerals

Beryl is the present commercial source of beryllium metal and beryllium compounds, which are used in ceramics, in the preparation of X-ray tubes and fluorescent lamps and screens, in special processes of paint and textile manufacture, and in the optical systems of some electrical instruments. The metal also is used in nuclear physics, chiefly as a source of neutrons. Beryllium is alloyed with aluminum for certain light-metal uses and is a constituent of some nickel and iron alloys. Currently the chief demand, however, is for copper-base alloys, which are exceptionally resistant to fatigue and wear, are responsive to hardening treatments after

RICHARD H. JAHNS, Member AIME, Professor, California Institute of Technology, Pasadena.

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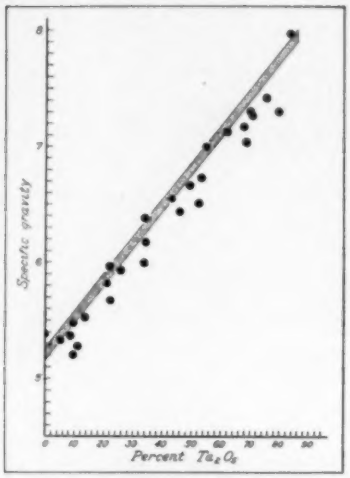

Fig. 2—Relation between specific gravity and tantalum oxide content in the columbite-tantalite series.

Stippled area represents theoretical range of values based on assumption that specific gravity of pure columbite is 5.20±0.05 and that of pure tantalite is 7.95±0.05. Black circles represent published analyses.

being worked soft, and are harder and otherwise superior to copper in structural characteristics. In addition, they are good electrical conductors, nonmagnetic, and nonsparking. Alloys of the beryllium-copper group are used, for example, in nonsparking tools and in springs, contact plates, bushings, shims, and corrosion-resistant parts of motors and gages. They also are employed for parts in precision instruments and machines.

Prices for clean-cobbed domestic beryl at the mine were quoted at $30 to $35 per short ton for several years prior to 1941, but they gradually rose during the succeeding wartime months until by May 1943 a price of $12 per dry short-ton unit was guaranteed by the Metals Reserve Co. for ore containing a minimum of 8 pct beryllium oxide (BeO). In June 1944, the price was raised to $14.50 per unit, but federal purchases were discontinued on Jan. 1, 1945. Subsequent open market transactions were carried on at distinctly lower price levels for a time, but prices since have risen to points well above the previous wartime high.

The BeO content of pure beryl varies with the proportion of certain alkalies present, notably sodium and cesium and, in general, ranges from less than 10 pct to a theoretical maximum of about 14 pct. Although satisfactory analytical methods have been developed for the determination of beryllium in ores,[15] they are time-consuming and expensive. Accurate analysis of low-grade material is particularly difficult. A fairly rapid qualitative method for recognition of BeO in minerals has been devised[16] and is reliable for material containing 1 pct or more of the oxide. An improved photometric method of analysis also has been reported,[15] and spectrochemical analysis, which is rapid and only moderately expensive, has been used successfully for numerous BeO determinations, especially in low-grade ores.[18] Dr. W. T. Schaller of the Geological Survey has been investigating the relations between indices of refraction, which are easily determined, and the BeO content of beryl, and this work promises to yield a rapid and inexpensive means for approximate analysis.

The beryl produced in the southwestern and southeastern United States is a byproduct derived from deposits of feldspar, lithium minerals, mica, or other pegmatite constituents, and hence the general output ordinarily varies according to market conditions for these other minerals. The beryl is recovered from the host pegmatite by rough cobbing and hand sorting, so only deposits containing coarse crystals and masses are of current economic interest. Several laboratory investigations, however, have demonstrated that at least some pegmatite containing small and scattered masses of beryl can be milled with reasonably good recovery of the mineral in the form of marketable concentrates.[17, 18]

The Black Hills area of South Dakota is the leading source of domestic beryl, and small quantities are obtained at irregular intervals from deposits in at least 14 additional states. The bulk of the beryl consumed in the United States, however, is imported from South American and other countries.

Other beryllium-bearing minerals, such as chrysoberyl, gadolinite, and phenakite, occur in many pegmatites, both in this country and abroad, but no deposits of present commercial importance are known. Phenakite, with a BeO content of about 45 pct, is a

Table I. Composition of Principal Beryllium Minerals[a]

Mineral	Simplified Formula	Theoretical Content of BeO, Pct
Beryl	$Be_3Al_2(SiO_3)_6$	13.97
Phenakite	Be_2SiO_4	45.45
Chrysoberyl	$BeAl_2O_4$	19.71
Helvite[b]	$R_4Be_3Si_3O_{12}S$	13.58 to 13.52
Gadolinite	$Be_2FeY_2Si_2O_{10}$	10.69
Beryllonite	$NaBePO_4$	19.69

[a] Includes minerals of present and potential commercial importance.

[b] Mineral group, comprising three theoretical end members: Helvite, $Mn_4Be_3Si_3O_{12}S$; Danalite, $Fe_4Be_3Si_3O_{12}S$; Genthelvite, $Zn_4Be_3Si_3O_{12}S$.

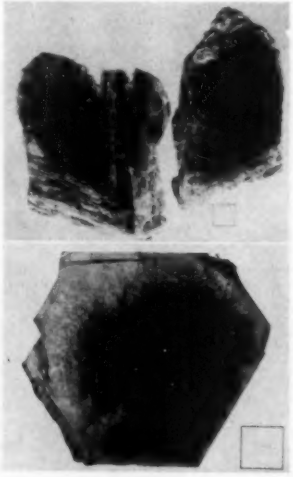

Fig. 3—Cleavage pieces of muscovite.

Top—Pieces marred by "A" reeves and cracks. Bottom—Flat piece with abundant hematite inclusions in form of flattened trigonal lattices. (One-inch square shows scale.)

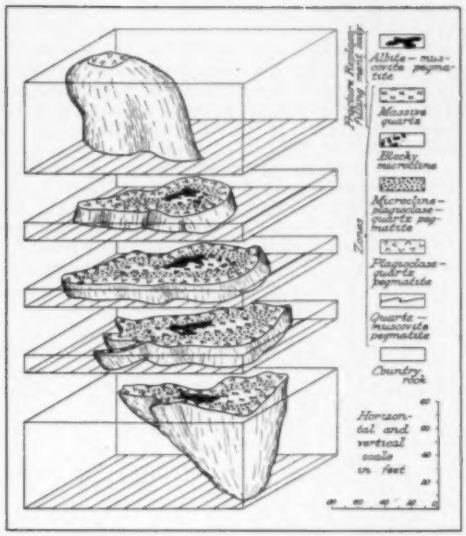

Fig. 4—Isometric block diagram of a typical steeply plunging podlike pegmatite body showing distribution of zones and other units at different levels.

locally abundant constituent of the Morefield pegmatite, Amelia County, Va. A reserve of several tons of this mineral might well be present, but most of the phenakite masses are too small to be recovered by ordinary hand sorting. Helvite and danalite, complex beryllium-bearing silicate minerals[10] occur abundantly in the Iron Mountain district, N. Mex. The deposits are of contact metamorphic rather than pegmatitic origin. Most are low in grade, and the difficulties involved in their metallurgical treatment appear to preclude their successful competition with beryl ores as a current source of beryllium.[10] The composition of principal beryllium minerals is given in Table I.

Lithium Minerals

Salts of lithium are used in pharmaceuticals, storage batteries, flares and fireworks, fluxes and special lubricants, as well as in the curing of meat, manufacturing of textiles and ceramics, refining of metals, smelting of iron ore, purifying of helium,[11] and dehumidifying of air in air-conditioning equipment. Lithium hydride has been employed as an effective transporter of hydrogen in self-inflating rafts, balloons, and other devices, and the metal is a minor constituent of some special-purpose alloys. The chief raw materials used for the manufacture of lithium compounds are the pegmatite mineral spodumene and the lithium-bearing brines of several saline lake deposits. Other pegmatite source minerals include amblygonite, zinnwaldite, and triphylite, but neither these nor spodumene contain large proportions of lithium or lithium oxide, as shown in Table II.

Spodumene can be employed directly in the manufacture of glass to neutralize shrinkage during cooling and also is an ingredient of some ceramic mixes. Lepidolite, the lithia mica, is much more widely used in glass making than as a source of lithium salts and is commercially as important as spodumene. Not only is it an excellent fluxing material, but it increases the luster, weather resistance, and strength characteristics of glasses, while reducing their coefficients of expansion. Such glasses are in great demand for high-pressure gages, electronic tubes, and other devices subject to mechanical stresses or sudden temperature changes. Lepidolite also is an ingredient of many high-quality porcelains and enamels in which it is an effective opacifier.

The average price quoted for spodumene during recent years has been about $30 per short ton of clean cobbed or milled material, generally with a minimum acceptability limit of 6 pct contained Li_2O. Prices for amblygonite, which contains a higher proportion of lithium, ordinarily have ranged from $40 to $50 per short ton. Lepidolite containing 3 pct Li_2O generally has commanded a price of $15 to $27 per short ton at the mine. Spodumene was allocated to consumers during the wartime period Dec. 5, 1942, to Sept. 30, 1944, when foreign supply was uncertain and domestic production facilities were not adequate in terms of demand. During this time prices rose to $50 or more per ton.

Most domestic spodumene has been obtained from deposits in North Carolina, South Dakota, and New Mexico. A group of pegmatites in the Kings Mountain area of the Carolina Piedmont constitutes by far the greatest domestic reserve of the mineral. This reserve may well amount to several million tons. Amblygonite is mined from time to time in

Table II. Composition of Commercial Lithium Minerals from Pegmatites

Mineral	Simplified Formula	Theoretical Content of Li_2O, Pct	Proportion of Li_2O in Most Marketed Concentrates, Pct
Spodumene	$LiAlSi_2O_6$	8.03	>6.0[a]
Lepidolite	$KLi_2AlSi_4O_{10}F_2$[b]	7.65[b]	>2.5
Amblygonite	$LiAlFPO_4$	10.10	>8.0
Zinnwaldite	$LiKFeAl_2F_2Si_2O_{10}$	3.40	>1.5
Triphylite	$LiFePO_4$	9.47	>3.5
Lithiophilite	$LiMnPO_4$	9.52	
Petalite	$LiAlSi_4O_{10}$	4.89	>3.0

[a] Concentrates containing as little as 4.0 pct Li_2O are sometimes marketed.
[b] For polylithionite end member.

Table III. Composition of Principal Tantalum-Columbium Minerals[a]

Mineral	Simplified Formula	Theoretical Content of Ta_2O_5 or Cb_2O_5, Pct	Proportion of Ta_2O_5 or Cb_2O_5 in High-Grade Concentrates, Pct
Tantalite[b]	$(Fe,Mn)Ta_2O_6$	86.12 Ta_2O_5[c]	>70 Ta_2O_5
Columbite[b]	$(Fe,Mn)Cb_2O_6$	76.88 Cb_2O_5[c]	>65 Cb_2O_5
Microlite[b]	$(Na,Ca)_2(Ta_2O_6)(O,OH,F)$	>82 Ta_2O_5	>70 Ta_2O_5
Pyrochlore[b]	$NaCaCb_2O_6F$	73.05 Cb_2O_5	>60 Cb_2O_5
Hatchettolite	$(Na,Ca,U)_2(Ta,Cb)_2O_6(O,OH,F)$	>65 $Ta_2O_5 + Cb_2O_5$	>60 $Ta_2O_5 + Cb_2O_5$
Samarskite	$(Y,Er,Ce,U,Ca,Fe,Th)(Cb,Ta)_2O_6$	>55 $Cb_2O_5 + Ta_2O_5$	>45 $Cb_2O_5 + Ta_2O_5$
Tapiolite[d]	$(Fe,Mn)(Ta,Cb)_2O_6$	86.01 Ta_2O_5	>70 Ta_2O_5
Fergusonite[b]	$(Y,Er,Ce,Fe)CbO_4$	>54 Cb_2O_5	>50 $Cb_2O_5 + Ta_2O_5$
Formanite[b]	$(Y,Er,Ce,Fe)TaO_4$	>66 Ta_2O_5	
Euxenite[e]	$(Y,Ca,Ce,U,Th)(Cb,Ta,Ti)_2O_6$	>55 $Cb_2O_5 + Ta_2O_5$	

[a] Includes minerals of present and potential commercial importance.
[b] End member of a tantalian-columbian mineral series.
[c] For material with Fe:Mn ratio of 1:1.
[d] Tantalum-columbium mineral series that include high-tantalum members but probably no corresponding high-columbium members.
[e] General name for members of a columbium-tantalum-titanium mineral series that are relatively rich in Ta and Cb.

Fig. 5—Harding quarry, Taos County, N. Mex. This view of the south wall shows the upper half of the gently dipping pegmatite dike. Note the well-defined layering.

South Dakota, and lepidolite has been recovered recently from deposits in South Dakota, Colorado, and New Mexico. Both minerals were obtained from the Stewart mine in southern California prior to 1926, and production of lepidolite from this mine was large.

Some lithium minerals are imported from Southwest Africa, Argentina, Brazil, Sweden, central and southwestern Europe, Australia, and Canada, but during most years the consumption of such materials in this country does not greatly exceed domestic production.

Recovery of lithium minerals from the host pegmatite ordinarily is a simple matter, involving hand cobbing and picking. Many crystals or crystalline aggregates of spodumene, lepidolite, or amblygonite are so large that they can be broken and shipped without sorting. In other places these minerals are intimately intergrown and can be marketed as mixed lithium-bearing material. Lithium minerals have been recovered by milling in at least three places during recent years. The flotation plant of the Solvay Process Co. near Kings Mountain, N. C., yielded commercial spodumene concentrates during the period 1943 to 1945.

Tantalum-Columbium Minerals

The metals tantalum and columbium are derived mainly from members of the tantalite-columbite series and to an appreciable extent from microlite and other minerals, see Table III. Columbium is alloyed with nickel, copper, and aluminum. Columbium-bearing alloy steels, with their favorable welding characteristics and high-temperature strength properties, are in demand for turbine and aircraft engine parts. Tantalum metal is used in radio and neon tubes, where its gas absorption properties are important, and its chemical inertness makes it unusually suitable for instruments and equipment that are exposed to corrosive liquids and fumes. It is employed in the manufacture of synthetic rubber and is uniquely satisfactory as a surgical metal. Tantalum is alloyed with columbium and tungsten to form dies and cutting tools, which also are made with cemented tantalum carbides. Tantalum-bearing glass is used in special camera lenses and other optical equipment.

The most desirable ores are microlite, low-tantalum columbite, and low-columbium tantalite that contain little tin or titanium. Columbium ore was valued at \$0.22 to \$0.40 per lb, and tantalum ore at \$1.80 to \$2.75 per lb in the years preceding World War II, but prices rose much higher as a result of wartime demands after 1941. The highest prices paid by the Metals Reserve Co. were in effect during the period July 1943 to January 1945, after which most purchases were discontinued. They ranged from \$2.20 per lb of contained tantalum oxide for concentrates assaying 40 pct Ta_2O_5 to \$4.65 for 75 pct concentrates, see Table IV. Columbite containing 50 pct Cb_2O_5 or more was purchased at \$0.50 per lb of contained oxide.

Accurate determination of grade is accomplished by chemical analysis, a slow and costly process, but rough estimates for members of the tantalite-columbite series can be made on the basis of the consistent relation between specific gravity and Ta_2O_5 content. In general, the specific gravity rises with increase in proportion of tantalum oxide as shown in Fig. 2. The specific gravity of most columbite ranges from 5.2 to 6.5 and that of all material commercially marketable as tantalite is greater than 6.5. Determinations can be made easily and quickly by means of gravity balances, which are available in many laboratories. Preliminary investigations of microlite from several deposits in the United States suggest that a similar relation holds for varieties of this mineral, with a general rise in specific gravity from 5.5 for material containing less than 60 pct Ta_2O_5 to as much as 7.0 for very high grade material.

Tantalum-columbium minerals occur in many pegmatite districts of the United States, but the

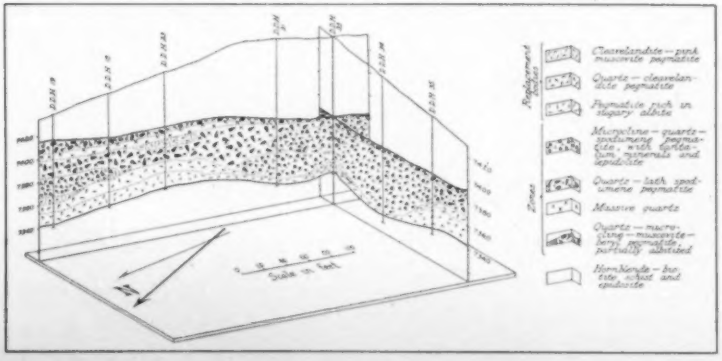

Fig. 6—Isometric plate diagram of a part of the Harding pegmatite, Taos County, N. Mex. See Fig. 9 for locations of diamond-drill holes.

average annual output has amounted to little more than 6000 lb during the past 30 years. Crystals and irregular chunks of tantalite and columbite are recovered from time to time as hand-cobbed byproducts from operations for feldspar and other pegmatite minerals, but very few pegmatites have been mined for tantalum-columbium minerals alone. Notable exceptions are the Harding pegmatites of northern New Mexico, which during World War II yielded the bulk of domestic tantalum production in the form of microlite concentrates. Additional microlite has been obtained since 1942 as a byproduct from other pegmatites in New Mexico, Colorado, and South Dakota.

Nearly all tantalum-columbium ore consumed in the United States is tantalite-columbite obtained from deposits in other countries, mainly Africa, South America, and Australia. Much of the highest-grade tantalite occurs in Australia and parts of South America. During recent years some tantalum was brought into this country in the form of smelter slags derived from central African tin ores, but this material contained very low percentages of Ta_2O_5. Other tantalum-bearing minerals such as microlite, hatchettolite, samarskite, and fergusonite, are widespread in their occurrence but are so sparsely distributed in most tantalum-bearing pegmatites that they ordinarily are of little economic interest.

Sheet Muscovite

The uses of sheet muscovite are based upon its perfect cleavage, remarkably low conductivity of heat and electricity, high dielectric strength, noninflammability, mechanical strength, flexibility, elasticity, transparency, luster, and the ease with which it can be worked into final form. The degrees of emphasis placed upon given properties by purchasers depend upon the specific end uses involved.[23-24] Flexibility is particularly important, for example, in the "cigarette" mica used in spark plugs for aircraft engines. This material, in films twelve ten-thousandths of an inch or less thick, is wrapped around rodlike spindles a little more than ⅛ in. in diam.[24] Condenser mica, in contrast, is valued because of its dielectric properties, and the use of mica for windows in furnace walls and doors is founded upon its transparency, heat resistance, and mechanical strength.

A very high proportion of sheet mica is used as an electrical insulating material. Washers, disks, and other small trimmed or stamped forms are not only employed as such, but they can be built up into

Table IV. Price Schedule for Domestic Tantalite Concentrates, July 1943 to January 1945[a]

Proportion of Ta_2O_5 in Ore, Pct[b]	Price per Lb of Contained Ta_2O_5, $	Proportion of Ta_2O_5 in Ore, Pct[b]	Price per Lb of Contained Ta_2O_5, $	Proportion of Ta_2O_5 in Ore, Pct[b]	Price per Lb of Contained Ta_2O_5, $
40	2.20	52	3.04	64	3.88
41	2.27	53	3.11	65	3.95
42	2.34	54	3.18	66	4.02
43	2.41	55	3.25	67	4.09
44	2.48	56	3.32	68	4.16
45	2.55	57	3.39	69	4.23
46	2.62	58	3.46	70	4.30
47	2.69	59	3.53	71	4.37
48	2.76	60	3.60	72	4.44
49	2.83	61	3.67	73	4.51
50	2.90	62	3.74	74	4.58
51	2.97	63	3.81	75	4.65

[a] Purchases made by Metals Reserve Co. at designated depots; schedule applies to lots in excess of 200 lb dry weight.
[b] Minimum acceptable Ta_2O_5 content is 40 pct; maximum allowable contents of SnO_2 and TiO_2 are 3 pct.

Fig. 7—Specimens of tantalum-lithium ore from the Harding pegmatite, Taos County, N. Mex.

High-grade ore. Aggregates of microlite, lithia mica, and minor tantalite are enclosed by laths of spodumene. Dark strips in specimen at left are smoky quartz.

rods, tubes, or other articles by cementing them with shellac, glyptol, or a similar bonding medium. Simple and composite pieces are used, for example, as sleeves, studs, tubes, washers, bushings, laminations, and thin perforated plates in condensers, transformers, small heating elements, rheostats, fuses, incandescent bulbs, radio and electronic tubes, various types of coils, and in acoustic, X-ray, and other specialized equipment. Thin splittings are built up into mica board or applied as facing on paper, cloth, and other materials used in the manufacture of heater elements; commutators; boards, panels, and other mounting forms; parts of condensers; and many other electrical devices.

The properties most desired in sheet muscovite of superior quality are flatness, uniform splitting characteristics, reasonable hardness and flexibility, elasticity, transparency, freedom from inclusions of other minerals, and freedom from such structural imperfections as cracks, tears, pinholes, warps, and ripples, see Fig. 3. The first rough separation of mica generally is made at the mine, where obvious scrap material is separated from the better books. Adhering fragments of quartz, feldspar, and other foreign material are then cobbed away from those books that contain sheet mica of usable quality. Some of this rough cobbed or selected mine-run mica is sold as such to jobbers or manufacturers, but at most mines it is prepared further. The books are split or "rifted" by means of knives, generally into plates 3/16 in. or less in thickness. Most defective laminae removed during this rifting process are discarded as shop scrap, which is distinctly less impure than typical mine scrap.

After rifting, the ragged and broken edges of many plates are removed with the fingers, a process known as thumb trimming. This is an especially common practice in districts where much of the mica is marred by structural imperfections. Some thumb-trimmed material is sold to manufacturers as such, but most is further trimmed with knives or shears and its value thereby increased. During recent years attempts have been made in the United States to employ several forms of blades and saws for semi-mechanized mica trimming, but without much success.

A large proportion of sheet mica is consumed in the form of splittings. These are films 0.0007 to 0.001 in. thick that generally are cleaved from the smaller sizes of sheet stock. Some also are derived from thin films or skimmings that are accumulated during the rifting of larger sheet material. Splittings are used in the manufacture of built-up mica board and other forms of electrical insulation. Although many mechanical devices have been tested for their preparation, most splittings are still made outside the United States by hand methods, generally in places where labor costs are very low.

The cut mica blocks that represent sheet material are further processed into disks, washers, and thin plates of various sizes and shapes. This processing generally involves additional splitting, followed by trimming, cutting, punching, or stamping into more or less standardized patterns. Most of this material is then cut to final form, if necessary, by the manufacturers of the devices in which the mica is to be used. Composite forms can be built up to any desired thickness by the cementing together of individual pieces.

In general, only a small proportion of the rifted and trimmed sheet mica is represented in the finished product. The bulk of such material is trimmed or cut away as waste, which is then marketed as scrap of superior grade.

Heavy demands for sheet mica of good quality are made during modern wartime periods. Such mica is used, for example, as splittings in the form of built-up mica commutator segments and coil insulations for motors and generators, and in transformers, switchboards, blasting apparatus, and aircraft generators and sparkplugs.[25, 26] In addition, an unprecedented problem was caused during World War II by the demands and exacting requirements for condenser mica. Developments in the field of military radio and electronic equipment focused attention on mica condensers, owing to the constancy and excellency of their electrical properties under varying physical conditions.[25]

Some of the problems encountered in increasing the domestic supply of so-called strategic sheet mica have been described and discussed in recent literature.[1, 25, 26-28] To stimulate production of such mica, the Metals Reserve Co., a subsidiary of the Reconstruction Finance Corp., designated the Colonial Mica Corp. as its agent, with authority to purchase mica of certain types and to assist the operators of mines with equipment leases, development loans, and consulting services on problems of mica mining and preparation. A market for mica of superior quality was assured at favorable prices and for specified periods.

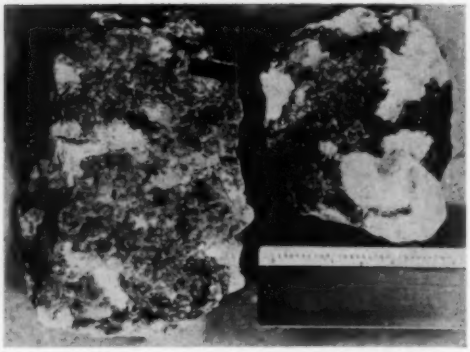

Fig. 8—Specimens of tantalum-lithium ore from the Harding pegmatite, Taos County, N. Mex.

"Spotted rock." Irregular masses of spodumene (white) are enclosed by an aggregate of microcline, albite, and minor tantalum minerals in specimen at left, and by purple lepidolite in specimen at right.

Table V. Price Ranges of Clear and Stained Sheet Mica in the Southeastern States During the Period 1910-1940

Size, In.	Range in Price, $ per Lb	
	Clear	Stained
1½x2	0.12 - 0.60	[a]
2x2	0.22 - 1.05	0.06 - 0.40
2x3	0.38 - 1.45	0.10 - 0.60
3x3	0.58 - 2.00	0.15 - 1.25
3x4	0.78 - 2.30	0.30 - 1.50
3x5	0.95 - 2.70	0.48 - 1.75
4x6	1.75 - 3.65	0.70 - 2.25
6x8	2.25 - 7.25	1.25 - 2.50
8x10	3.50 - 11.50	2.00 - 3.00

[a] Under ordinary conditions the smallest size of stained mica purchased as sheet material in the Southeastern States is 2x3 in.

Table VI. Price Schedules for Domestic Clear Sheet Mica During Period December 1941-February 1945[a]

Size, In.	Prices, $ per Lb						
	Private Purchasers	Colonial Mica Corp.					Private Purchasers
	April-May 1942	June 1942[b]	November 1942[b]	May 1943[c]	February 1944[c]	August 1944	February 1945[f]
Punch	0.12 - 0.16	0.22	0.30				0.08 - 0.15
1½x2	0.50 - 0.65	1.10	2.40			6.00[d]	1.00
2x2	0.95 - 1.10	1.75	3.52				1.40
2x3	1.50 - 1.85	2.75	4.64				2.00
3x3	2.00 - 2.35	3.50	5.12	5.00	6.00	2.25[d]	2.55
3x4	2.25 - 2.60	4.25	6.08			6.00[e] and 8.00[d]	3.00
3x5	2.75 - 3.00	5.00	7.04				3.45
4x6	3.75 - 4.00	6.25	8.00				4.30
6x8	5.50 - 6.00	8.00	9.12				6.50

[a] Adapted in part from Billings and Montague. The Wartime Problem of Mica Supply: *Engineering and Mining Journal* (1944) 145, p. 94.
[b] Punch material required to yield 20 pct or more of trimmed pieces 1x1 in. or larger; price scale for larger mica based on No. 1 quality and half trim, with max bonuses of 30 pct and 40 pct for three-quarter trim and full trim, respectively.
[c] Based on full-trimmed punch and three-quarter trimmed sheet mica.
[d] Full-trimmed mica.
[e] Three-quarter trimmed mica.
[f] Based on untrimmed punch and three-quarter trimmed sheet mica.

Prices for sheet mica not only fluctuate widely in response to variations in demand, but vary at any given time according to the size and quality of the material involved. The ranges for clear sheet mica are great, and during a 30-year period the price for 3 x 3 in. trimmed sheets in the southeastern United States, for example, ranged from $0.58 to $2 per lb, and that for 1½ x 2 in. trimmed sheets ranged from $0.12 to $0.60 per lb as shown in Table V. During war periods, when demands are greatly increased and problems of supply often are complex, prices characteristically reach very high levels. Thus the trend rose rapidly during the period December 1941 to December 1944, until a flat price of $8 per lb for full-trimmed sheets 2 x 2 in. and larger was in effect, see Table VI. Subsequent price scales have been considerably lower and again have been based upon size and quality of the prepared sheets.

Most of the world's sheet muscovite is obtained from India, and much smaller quantities are recovered from deposits in Brazil, Argentina, the United States, Russia, and several other countries. Domestic production rarely amounts to more than 20 pct of consumption, and during most war periods this proportion decreases distinctly. Exhaustive tests have shown that there is little intrinsic difference in electrical and other physical properties between American and Indian muscovites,[26, 28] but much of the imported material is consistently better prepared and more carefully graded.

Pegmatite Deposits

Most pegmatites can be classified as sills or dikes, depending upon whether or not they are conformable with the structure of the enclosing country rock. Variations of these forms include markedly pinching-and-swelling bodies and series of disconnected lenses and pods. In addition, troughlike, funnel-like, cigar-shaped, mushroom-shaped, and various branching forms have been recognized. Studies in all parts of the United States have shown that, despite numerous complexities of detail, most pegmatite bodies of commercial interest are rather regular in general structure. Plunging bodies are especially common, and many pegmatites that appear to be simple sills or dikes actually are shaped in three dimensions more like laths or flattened cigars, with long axes that plunge gently to moderately. Such plunging structures are extremely important as far as economic exploration of the deposits is concerned.

Pegmatite bodies that cannot be divided readily into units of contrasting composition and texture appear to constitute the great bulk of pegmatitic material in some areas. In general, however, these have received much less attention than pegmatites that are lithologically and structurally more complex. This latter group includes nearly all pegmatites that contain rare minerals, as well as most of those with minable concentrations of feldspar, mica, beryl, and other minerals. A general systematic arrangement of lithologic units in such pegmatites has long been recognized, and bands, barrels, columns, layers, lenses, pipes, pods, ribs, shoots, streaks, veins, and zones are terms commonly used by miners and referred to in geologic literature.

An essentially regular and orderly internal structure in many pegmatites was described and discussed by some earlier geologists, and even was shown on maps and sections by a few, but the attention of most investigators was directed more toward questions of mineralogy and genesis than toward structural considerations. It has remained for more recent investigators to place greater emphasis upon detailed mapping and structural interpretation of individual parts of pegmatite bodies and to demonstrate more fully the economic value of such studies in prospecting and in the planning of exploration, development, and mining.[3, 4, 5, 7-12, 22-29] In many areas it has been shown by careful studies—and in places confirmed by subsequent mining—that concentrations of economically desirable pegmatite minerals commonly occur in rock units quite distinct from adjacent barren units. Moreover, the shape and dis-

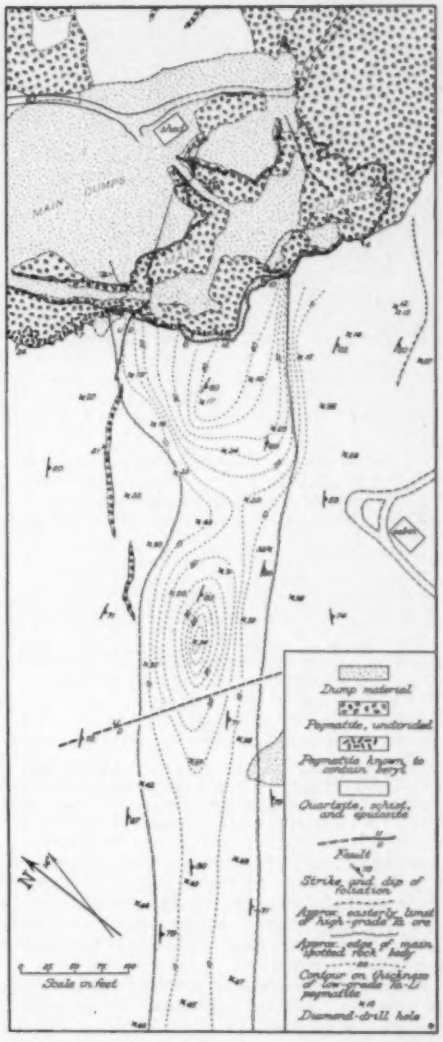

Fig. 9—Map of a part of the Harding mine area, Taos County, N. Mex., showing distribution of U. S. Bureau of Mines diamond-drill holes and principal mass of tantalum-lithium pegmatite.

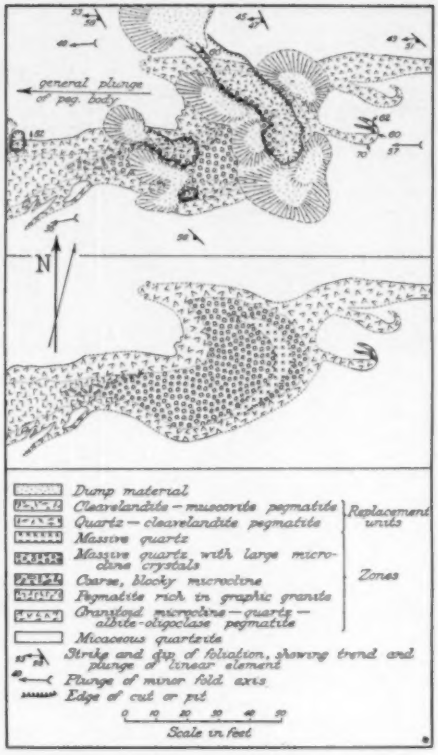
general plunge of peg. body
N
Dump material
Clearelandite – muscovite pegmatite
Quartz – clearelandite pegmatite
Massive quartz
Massive quartz, with large microcline crystals
Coarse, blocky microcline
Pegmatite rich in graphic granite
Granofoid microcline – quartz – albite-oligoclase pegmatite
Micaceous quartzite
Strike and dip of foliation, showing trend and plunge of linear element
Plunge of minor fold axis
Edge of cut or pit
Replacement units
Zones
0 10 20 30 40 50
Scale in feet

Fig. 10—Geologic map of a part of the Pino Verde pegmatite, Rio Arriba County, N. Mex., with insert showing probable distribution of zones prior to albitization and development of other minerals by deuteric replacement.

tribution of many of these units reflect the general shape or structure of the host pegmatite body. The recent work thus has fully confirmed the observations and suggestions made by several earlier investigators who devoted attention to the internal structure of pegmatites.

According to a recently proposed classification,[11] the internal units of pegmatites comprise three fundamental types:

1. *Fracture fillings,* which are generally tabular bodies that fill fractures in previously consolidated pegmatite.

2. *Replacement bodies,* which are formed primarily by replacement of pre-existing pegmatite, with or without obvious structural control.

3. *Zones,* which are successive shells, complete or incomplete, that commonly reflect the shape or structure of the containing pegmatite body. Where ideally developed, they are concentric about an innermost zone, or core.

Typical pegmatite units are shown in Fig. 4. Such units range widely in size, shape, and texture. The smallest are tiny fracture-filling veinlets and the thin outermost zones of many pegmatite bodies, and the largest are masses several hundred feet long and more than 50 ft in minimum dimension. Many units are easily distinguished and sharply bounded from adjacent units, especially where they differ markedly from them in composition or texture. Contacts between others are gradational, and in some bodies of very coarse-grained pegmatite they are difficult to locate within narrow limits. Even where adjacent units are mineralogically similar, however, most boundaries can be mapped conveniently on scales of 20 or 25 ft to the inch, and independent assignments of such boundaries by more than one geologist generally agree within narrow limits.

The distribution, structure, and economic significance of many pegmatites and pegmatite units in the southwestern and southeastern United States can be most briefly explained by means of examples. The Harding pegmatites, Taos County, N. Mex., many of the pegmatites in the Petaca district, Rio Arriba County, N. Mex., and most of the commercial mica-bearing pegmatites of western North Carolina consist of two or more readily distinguishable units, and hence contain concentrations, rather than uniform disseminations, of commercially desirable minerals. The only noteworthy exceptions are in North Carolina, where much mica has been recovered from thin, rather homogeneous pegmatite lenses.

The Harding lithium-tantalum-beryllium pegmatites, in western Taos County, N. Mex., were worked during the period 1920 to 1930 for their lithium minerals, which were used in the manufacture of glass and ceramic products. Tantalum minerals were identified in several of the deposits soon after they were opened up for mining, but it was not until the early months of World War II, when Arthur Montgomery of New York City re-examined the mine openings in detail, that the abundance and commercial possibilities of these tantalum minerals were recognized. Mr. Montgomery began mining operations in December 1942, and since that time the Harding pegmatites have been the foremost domestic source of tantalum, with a total production of more than ten tons of concentrates containing an average of about 70 pct Ta_2O_5. The chief tantalum mineral is microlite, but appreciable quantities of tantalite-columbite and a little hatchettolite were included in the concentrates marketed. More than 500 lb of coarse tantalite-columbite, averaging 45 to 55 pct Ta_2O_5, was recovered from small placer deposits derived from the pegmatites.

The pegmatite of principal interest is a thick, nearly flat dike that lies within a zone of probable thrust faulting in steeply dipping metamorphic rocks of pre-Cambrian age. It dips southward, has an average thickness of 50 to 55 ft, and is known to contain beryllium, tantalum-columbium, and lithium minerals for distances of 350 ft along the strike and 650 ft down the dip. Its upper half is exceptionally well exposed along the face of a large quarry shown in Fig. 5, and its down-dip extensions were diamond drilled in 1943 and 1948 as a part of a program of thorough exploration by the U. S. Bureau of Mines.[14] This dike is strikingly layered, each layer consisting of a distinct rock type whose general position within the dike is remarkably consistent, although a given layer may show considerable variation in thickness. The layered structure has been a helpful and reliable guide in the prediction of ore-mineral distribution.

Most of the commercially recoverable beryl occurs in two 6-in. to 5-ft layers of quartz-microcline[a]-muscovite-albite pegmatite, one of which lies be-

[a] All microcline noted in this paper contains perthitic intergrowths of sodic plagioclase.

Fig. 11—Mica-bearing pegmatites, Rio Arriba County, N. Mex. Rich concentration of coarse, partly albitized mica books near pegmatite-wall rock contact, Joseph mine.

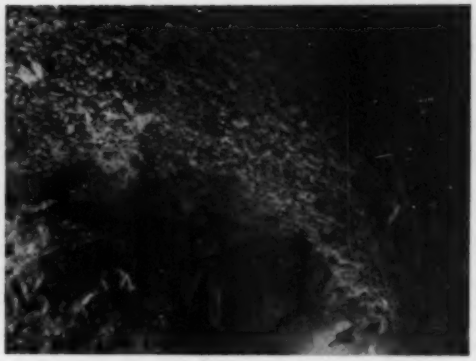

Fig. 12—Mica-bearing pegmatites, Rio Arriba County, N. Mex. Broad roll in hanging wall of Apache pegmatite. Quartz core in left foreground, remnants of mica-rich pegmatite in back of stope.

neath the hanging wall and the other above the footwall of the dike. They are best developed in its thickest parts. The beryl is white, pale yellowish green, and pinkish in color, and generally lacks crystal form. Some of the masses are extremely large, weighing 20 tons or more, and hence much of the material is readily recoverable by hand cobbing.

Immediately beneath the hanging-wall beryl-bearing layer is a somewhat thicker zone of massive quartz, which in turn grades downward into a thick zone of quartz with long, slender laths of spodumene that are spectacularly exposed on the quarry walls as shown in Fig. 5. Irregular masses of white to pinkish beryl, ½ in. to 2 ft in maximum dimension, are locally interstitial to the spodumene laths near the base of the exposed part of this unit. The beryl is associated with white to green microcline in a few places, but nowhere in the zone is either of these minerals as abundant as the quartz. The core of the dike is a coarse-grained aggregate of spodumene, microcline, and quartz, with varying quantities of albite, muscovite, lithium muscovite, lepidolite, and tantalum-columbium minerals. The pegmatite would have been symmetrically zoned at one time were not many of the original lithologic units in its lower half obscured or even obliterated by albitization, and locally through replacement by mica. The present distribution of rock types is markedly asymmetric, see Fig. 6.

The high-grade tantalum ore consists of small crystals and grains of yellow to brownish microlite and subordinate manganotantalite in a matrix of lithia mica, or locally in quartz, spodumene, muscovite, albite, or aggregates of these minerals as shown in Figs. 7 and 8. Small masses of such ore occur in a rather well-defined belt near the center of the dike. This belt appears to be 100 to 150 ft wide, and concentrations of tantalum minerals occur through a vertical range of about 15 ft. The distribution of rich ore is very irregular in detail but is surprisingly uniform within many large blocks of ground. Individual shoots have been mined by means of irregularly branching drifts, inclines, and low rooms.

Tantalite-columbite, yellowish brown to black microlite, and minor quantities of hatchettolite are disseminated throughout the central, spodumene-rich unit in the pegmatite body, see Fig. 9. This unit consists mainly of pinkish to gray, coarse but even-grained pegmatite to which the name "spotted rock" has been applied, as shown in Fig. 8. Most of the spodumene masses are nearly equidimensional, in contrast to the spodumene laths in the zone immediately above, and they average about 1 in. in diam. They are associated with microcline, albite, and quartz. Replacement of the potash feldspar and spodumene of the "spotted rock" by lepidolite has yielded large bodies of pegmatite rich in the lithia mica. The aggregates of this mica contain numerous remnants of the earlier minerals, showing all stages of replacement. Several of the lepidolite concentrations were mined out during the period 1920 to 1930.

In general the spotted rock, or low-grade tantalum-lithium-bearing pegmatite, lies immediately west of the high-grade microlite shoots. The shape of this large rock unit, as determined by observations of surface exposures and diamond-drill cores, appears to be similar to that of a loaf of French bread, with one prominent constriction and an equally prominent vertical bulge as shown in Fig. 9. Its long axis plunges 2° to 7° SW, and thus rakes slightly westward down the gentle southerly dip of the dike. The mass is about 175 ft wide and 25 to 40 ft thick, with a long, or down-plunge, dimension of at least 750 ft from the original outcrop. Its position is reflected in some places by a broad bulge in the hanging wall of the dike, and in others by a sag in the footwall, see Fig. 6.

Although much of the tantalum and some of the beryllium and lithium minerals apparently were developed by deuteric replacement of pre-existing pegmatite, their vertical distribution is broadly controlled by the layered or zonal structure of the dike. Moreover, their along-strike and down-dip distribution is in accord with discontinuities and changes in thickness of the zones. A few small fracture-controlled replacement bodies of albite and quartz with varying quantities of beryl, microlite, tantalite, hatchettolite, and bismuth minerals transect zones and zone boundaries at distinct angles, but these bodies are not abundant and are of little economic significance.

Pegmatite deposits in the Petaca district of Rio Arriba County, N. Mex., have been sources of commercial muscovite since the seventeenth century, and some of the mines may well represent the oldest systematic operations for sheet mica in this country.

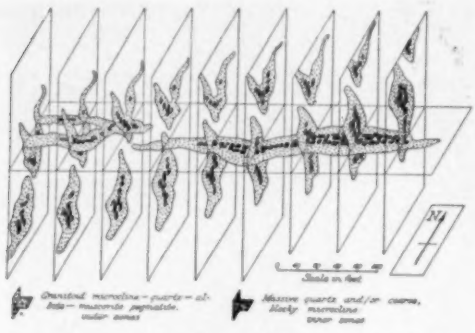

Fig. 13—Isometric plate diagram of a typical dike and a typical troughlike pegmatite body, Petaca district, N. Mex.

These and similar deposits elsewhere in northern New Mexico have been investigated in detail during recent years by the U. S. Geological Survey, U. S. Bureau of Mines, and other organizations.[4, 12, 16–17]

The Petaca pegmatite bodies include dikes, sills, troughlike and funnel-like masses, and masses of more irregular shape, and are abundant in an elongate area of pre-Cambrian schist, quartzite, metarhyolite, and granite.[4] Most of the pegmatites are discordant, with strikingly uniform westward trends, steep dips, and moderate westward plunges. These plunges are consistently conformable with those of numerous minor structural elements in the adjacent country rock, mainly the axes of drag folds, intersections of bedding and foliation planes, and axes of "stretched" phenocrysts in the metavolcanic rocks and tectonically elongated pebbles in several conglomerate beds.

Microcline and quartz, the chief pegmatite minerals, commonly form easily distinguishable zones within individual pegmatite bodies. In general, cores of massive quartz are surrounded successively by zones of coarse, blocky microcline or of massive quartz with euhedral to subhedral microcline crystals 2 ft or more in diam, and by outer zones of fine to coarse-grained microcline-quartz pegmatite with granitoid texture as shown in Fig. 10. The border selvages of a few deposits contain albite-oligoclase. Some of the cores can be traced along their strikes into veinlike masses of quartz, and offshoots from others transect the enclosing zones and continue into the country rock as thinly tabular masses.

Superimposed upon the nearly concentric structural pattern of the zones are fracture fillings and fracture-controlled replacement bodies of quartz, cleavelandite, muscovite, and other minerals, either singly or in combinations, see Fig. 10. An association of book muscovite with albite, particularly the variety cleavelandite, is strikingly prevalent throughout the district. Little mica is present in deposits or parts of deposits that contain little albite, whereas all deposits rich in mica are also rich in the late-stage soda feldspar. The amount of muscovite in a given pegmatite unit generally is proportional to the amount of albite, but this relation does not always hold where the pegmatite consists almost entirely of albite. Such albite-rich units may contain the remnants of many mica books that appear to have been attacked and partly replaced by the plagioclase, see Figs. 11 and 12. There seems to be no correlation between the size of a given pegmatite body and its mica content. Both large and small pegmatites contain rich and extensive mica shoots, yet some of the largest dikes in the district are relatively barren of mica.

Concentrations of albite and mica are present in deposits of all general forms. They are most abundant at the keels and along the lower flanks of plunging dikes, and hence appear near the eastern ends of such bodies at their outcrops as shown in Fig. 13. They also are common at and near the keels of plunging troughlike bodies, which are U-shaped, hook-shaped, or boomerang-shaped in plan. The muscovite is thus most abundant at and near the main bends, which generally constitute the easternmost exposed parts of such pegmatites. On the other hand, the mica in plunging inverted troughlike bodies is most abundant near their western exposed parts and so is concentrated near and along their crests. In other less regular bodies, mica shoots commonly occur along bulges, indentations, "rolls," or junctions with branches, see Figs. 11 and 12.

The shapes of the mica shoots characteristically reflect the shape and structure of the enclosing zones, although in detail many shoots transect zone boundaries at distinct angles. In this latter respect the Petaca deposits are fundamentally different from most of the mica-bearing pegmatites of New England and the Southeastern States in which commercial concentrations of muscovite are largely restricted to specific zones and appear to have been developed earlier during formation of the zones.

Mica-bearing pegmatite has been mined extensively in western North Carolina for nearly 50 years, and this region has accounted for more than half the total domestic production of sheet and punch mica during that time. At least 3300 deposits are known to have been worked, and during the recent war period substantial quantities of mica were obtained from more than 1200 deposits. The Spruce Pine district, which occupies parts of Avery, Mitchell, and Yancey Counties in the Blue Ridge province, is the largest in North America, both in total production of mica and in the number of mines and prospects that lie within its limits. Since 1900 the output from at least 800 mines in this district has been marketed, and hundreds of other deposits have been prospected.

The pegmatites in North Carolina and adjacent states have been studied in considerable detail during recent years, chiefly by the U. S. Geological Survey and the U. S. Bureau of Mines.[6, 8, 9, 20, 22, 30–42] Most of the mica-bearing pegmatites occur in a metamorphic terrane that may be pre-Cambrian in age. The country rock comprises mica schist and

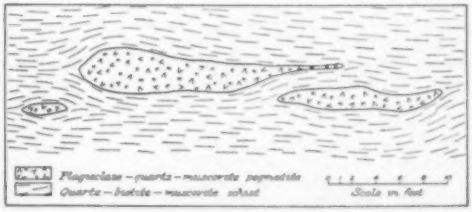

Fig. 14—Lenses of muscovite-bearing pegmatite in which zonal structure is not well developed.
Sketched from wall of stope in Elk mine, Avery County, N. C.

gneiss, impure quartzite, and hornblende schist and gneiss, with minor interlayered kyanite gneiss, sillimanite gneiss, graphitic schist, recrystallized limestone and dolomite, and various types of chloritic rocks. Large masses of silicic intrusive rocks, probably late Paleozoic in age, are exposed in many areas. They range in composition from quartz monzonite to quartz diorite, and commonly are surrounded and locally transected by somewhat finer-grained satellitic sills and dikes of similar composition. Most of the pegmatites that contain commercial concentrations of muscovite also are similar in composition to the large intrusive masses, and in some areas they are demonstrably related to them in genesis. Some of the pegmatites occur within, rather than adjacent to, the larger igneous masses, but they are in the minority and account for only a small proportion of the mica produced in the State.

The pegmatites are granodioritic rather than truly granitic in composition and consist of plagioclase and quartz with subordinate microcline, muscovite, biotite, and accessory minerals. Muscovite is present in some deposits as disseminated flakes, foils, and tiny books, but in others it occurs as very large books, a few of which are as much as 2 ft in diam and weigh several hundred pounds. All variations between these extremes are known, and many pegmatites contain book muscovite in a wide variety of types and forms. In general, however, the concentrations of muscovite occur within certain zones and are restricted to those zones. Thus the distribution of mica within a given deposit reflects the shape of the containing pegmatite body, although there appears to be little correlation between the size of the body and the quantity of mica within it.

Some deposits of book muscovite in the Southeastern States occur in lenses, tongues, or other pegmatite bodies that are not clearly zoned. Few such bodies are more than 6 ft thick, and many are less than 1 ft thick, see Fig 14. In general, however, the mica deposits are in well-defined units that ordinarily are quite distinct from adjacent barren units. Their distribution is clearly governed by zonal structures. Entire zones in some pegmatites are sufficiently rich in mica to be mined, but most mica deposits are confined to certain parts of zones and hence occur as shoots not unlike the shoots of ore minerals in metalliferous deposits. The position and distribution of such shoots commonly can be correlated with the overall shape of the containing zones, or with rolls, bends, constrictions, bulges, protuberances, or other irregularities in the zones, see Fig. 15.

Only a small quantity of merchantable sheet mica is associated with fracture fillings or replacement bodies, particularly in the pegmatites of greatest output. This relation is in marked contrast to that in the pegmatites of New Mexico in which nearly all the book-mica concentrations were developed in part after formation of the respective pegmatite zones that contain them. Most of the sheet muscovite produced from the Southeastern deposits is obtained from zones near the pegmatite walls. Substantial quantities also are recovered from disseminated books in poorly zoned pegmatites, especially those that are thin and lenticular, and from concentrations along the margins of quartz cores or interior zones of coarse-grained quartz and microcline.

Detailed field studies have demonstrated that the mica within a given zone is rather consistent in color, clarity, type and distribution of structural

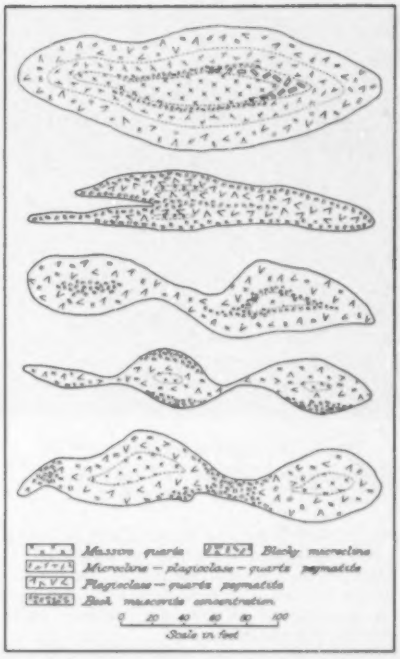

Fig. 15—Idealized plans of zoned North Carolina pegmatites showing characteristic distribution of book muscovite concentrations.

defects, and other physical properties, whereas the books from different zones within the same pegmatite commonly differ very strikingly. Green mica with numerous ridges and crenulations, or reeves, is especially abundant along the edges of quartz cores in many pegmatites, for example, whereas the mica in the earlier-formed wall zones of the same pegmatites is cinnamon brown, brown, or brownish olive, and is relatively free from structural defects. Severely reeved mica is most abundant in the pegmatites and pegmatite zones that are rich in potash feldspar, and books of the best quality generally are most numerous in oligoclase-rich zones.

Pegmatite Mining: Operations

Most pegmatite mining is done on a small scale, with crews rarely comprising more than six men. Methods of mining and types of mine workings vary according to the size, shape, attitude, and degree of weathering of the deposits, as well as the type of labor and mining equipment available at the time.

Nearly all mines are outgrowths of prospecting operations on promising surface exposures and thus represent enlargements of trenches, shallow pits, and small, irregular cuts. As prospect openings are deepened or extended horizontally in deposits rich enough to sustain mining activities, a change to underground operations commonly is made. This is particularly common in pegmatites with moderate or gentle dips, or in other deposits where further open-cut work would require excessive timbering or the removal of much overburden. Crosscuts, drifts, stopes, and other openings are developed from adits, shafts, and inclines as the pegmatite is mined. Some

Fig. 16—Mica mining in kaolinized pegmatite.

Large mica books in weathered wall zone of White Peak No. 1 pegmatite, Powhatan County, Virginia. Quartz core is exposed on left wall of cut, country-rock schist on right wall.

of these underground workings follow definite patterns, but others are developed by highly irregular and unsystematic mining operations known as gophering or jayhawking. Such tortuous openings are especially common in the near-surface parts of weathered pegmatites.

Methods of mining are characteristically simple. The soft, decomposed pegmatite of numerous areas in the Southeastern States, for example, is easily handled with a pick and shovel. Crystals of muscovite, beryl, and other minerals that are resistant to chemical decay can be picked out of the kaolinized host rock, as shown in Figs. 16 and 17, and the waste material then can be hoisted away in buckets or small skips, dragged away in scrapers, or trammed in wheelbarrows or cars. The general simplicity of such operations has underlain the miners' preference for working in soft rock wherever possible, despite the constant danger of caving and the need for timbering. Timbering ordinarily is kept to an absolute minimum, and many of the workings collapse soon after they are opened.

Ordinary quarrying methods are employed in the mining of many unweathered pegmatites, even in some places where relatively large quantities of barren rock must be handled to permit recovery of marketable minerals. Irregular drifts, stopes, and additional underground workings are developed from surface openings in other pegmatites, and still other deposits are reached by means of shafts, inclines, or adits driven through the adjacent country rock. Drilling was done by hand in many of the older pegmatite mines, but during recent years, especially during World War II, portable compressors and mechanical drills have been widely used. Power hoists, motor-generator sets, high-capacity pumps, and other types of modern equipment are also employed in numerous current operations.

Stripping methods have found increased favor during recent years, both for removing overburden from gently dipping deposits and for eliminating dangerously steep or overhanging walls of cuts. Highly mechanized techniques, involving the use of drag-line scrapers, bulldozers, and power shovels, have proved effective at several deeply weathered deposits. One of the most spectacular operations of this type resulted in the removal of enormous quantities of country-rock schist from the gently dipping, mica-rich Big Bess pegmatite in Gaston County, North Carolina, see Fig. 17. More than 70 ft of overburden was successfully removed from parts of this pegmatite in 1944 and 1945, and very little blasting was necessary. Similar operations in some mica-bearing Brazilian pegmatites appear to have yielded satisfactory returns."

Several weathered pegmatites in the Southeastern States have been mined by bulk methods, chiefly with bulldozers or drag-line scrapers. Mica and other minerals that had been left in pillars and walls of old, shallow underground mine workings were thereby recovered.

Production and Yields

The bulk of pegmatite mineral output in this country ordinarily is obtained from a relatively few mines at any given time. During the period 1942 to 1945, for example, only 83 mines in the Southeastern States qualified as very large, i.e., each of them yielded 3000 or more lb of trimmed punch and sheet muscovite. They represented only 4.5 pct of all mines and prospects in the Southeastern States from which production was obtained during that period, and yet their combined output amounted to more than 71 pct of the total from all the productive deposits as shown in Fig. 18.

In general the large and very large mica mines were operated at a profit during the period of wartime prices and other subsidies, and some of them have been satisfactory sources of commercial muscovite during other less favorable periods as well. Analyses of operating data indicate that most of these mines owe their productivity either to relatively high proportions of recoverable muscovite in the pegmatite handled, to relatively high proportions of sheet stock in the mine-run mica, or to combinations of these factors. This is in full accord with the results of similar analyses of the mica-producing districts in New England.[1] In commenting upon the yields of sheet mica from New England mines, Bannerman and Cameron[1] state that "there is no sharp division between rich and lean mines. Barring a price scale so low that no mines can operate, prices cannot be set that will eliminate the problem of marginal mines, and every revision of price during the war has brought a new group of mines into the marginal field."

The relatively large number of mica mines gives great statistical value to data bearing on their production and yields. The less abundant and widespread data that are available for beryllium, lithium, and tantalum-columbium-bearing pegmatites show similar relations. Only a few beryllium-bearing pegmatites, for example, furnish the bulk of domestic beryl production, and in general the productivity of such deposits is directly ascribable to their high yields of coarse, readily recoverable beryl per ton of rock mined. Similar relations appear to obtain in the case of other pegmatite groups, the only notable exception comprising the deposits from which relatively minor constituents are recovered as by-products.

Development of Reserves

Mining operations in pegmatite deposits have been cited repeatedly as poor financial risks, owing in large measure to uncertainties arising from the

smallness and irregularity of minable concentrations of marketable minerals. Pegmatite mining traditionally has been started on promising outcrops and has been continued only until the exposed concentrations of desirable minerals were worked out or until other conditions made operations unprofitable. Thus reserves rarely are developed in advance of actual mining, and individual operations are characteristically short-lived. Moreover, the selective mining of the richest parts of mica shoots and other mineral concentrations leads to development of tortuous gopherhole workings, as well as to the leaving of much valuable material in the ground. Not only are such methods plainly wasteful, but they impose limitations on the depth to which a given shoot can be worked effectively, and they fail to uncover reserves in the form of adjacent shoots. Uncertain market conditions, limitations of available capital, and the low margin of profit commonly obtained from the sale of many pegmatite commodities militate against aggressive exploration and development of reserves. Numerous attempts have been made and a few have been conspicuously successful, but unfortunately most have antedated the accumulation of adequate structural data on the deposit being tested. In the Petaca district of New Mexico, for example, several adits and shafts were aimed at down-dip extensions of known deposits, but were developed without knowledge of the gentle to moderate plunges of those deposits. Several low-level adits that were driven to intersect productive parts of pegmatite bodies thus passed entirely beneath the keels of these bodies. Although they actually emphasize the geometric significance of the plunges involved, such disappointments nevertheless have been mentioned repeatedly in support of the thesis that pegmatite bodies are too irregular to be developed in advance of mining.

It is no accident that the exploitation of rich mica concentrations in the Petaca district downward from their outcrops has led to the development of inclines and inclined stopes that generally slope in a westward direction. It is this raking, or down-plunge, direction of mica shoots and of the pegmatite bodies themselves that plainly should receive prime consideration in the planning of future exploration and development in the district. Comparable structural features exist in other pegmatite areas, and guides for prospecting commonly can be worked out by careful analysis of each pegmatite body involved.

Outlook

The rate of discovery of new pegmatite deposits in this country has dwindled markedly during the past few decades. During the period of World War II, when prospecting and mining activities reached an all-time high, the proportion of mica and most other strategic pegmatite minerals obtained from deposits newly discovered was smaller than ever before. With the steadily increasing need for accurate appraisal of pegmatites and pegmatite groups, it has therefore become more and more necessary to focus attention on mineral concentrations not exposed at the surface and upon those only partly mined out during earlier periods of activity.

Serious problems generally confront the operator who considers reopening a pegmatite mine. The workings of many mines are inaccessible, owing to flooding, caving, or fouling with tons of backfill. For many mines, maps are not available, and reports concerning the size and distribution of workings are incomplete or inaccurate. Production and cost data rarely are complete and reliable, and often deposits must be judged solely on their general reputation among local residents. Ordinarily it is difficult to determine whether previous mining was discontinued because of intrinsic shortcomings of the deposit, poor planning or inefficiency of operations, or market conditions, or because of other factors, and it is frequently impossible to contact former operators for discussion of such matters.

Fig. 17—Mica mining in kaolinized pegmatite.

Mechanized mining in the gently dipping Big Boss pegmatite, Gaston County, North Carolina. The pegmatite underlies the bench upon which the truck and bulldozer are standing.

Available surface exposures of pegmatites generally are little more satisfactory than mines for furnishing readily usable information. Few contain promising shoots of mica or other desirable minerals, and many are so small or discontinuous that they do not provide a clear picture of the pegmatite or its structure. Other outcrops, however, can be used to determine the internal structure of the pegmatite body and hence to determine the probable existence or distribution of mineral concentrations.

Despite the numerous difficulties involved, prediction of future possibilities for pegmatite mining is far from a hopeless task. The recent war period of high prices and subsidies led to an unprecedented expansion of pegmatite mining and prospecting in this country. Of necessity, much was done hastily and unsystematically, but opportunities for the study of pegmatites and pegmatite deposits in three dimensions were provided on a broader scale than ever before. It was possible to record and collate data on internal structure of the deposits, lithologic sequences in certain kinds of pegmatite bodies, types of book mica and other minerals, and many other features of direct or indirect economic application. More maps and detailed sections were obtained during the short period of wartime activity than had been accumulated throughout all preceding periods. The advantages of such maps and diagrams in planning and executing future operations are evident.

After they were opened during World War II, some pod and lenslike pegmatites were found to have been essentially mined out during previous periods of activity, but the mine workings in a great many other pegmatites are now known to be short of the limits of workable mineral concentrations.

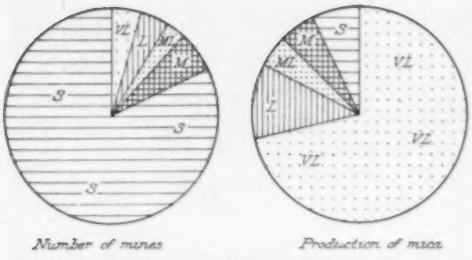

Fig. 18—Classification of North Carolina mica mines according to their yields of sheet muscovite, showing the marked contrast between the relative number of small mines and their relative aggregate output.

Classification of mines, based on production during World War II:

VL — Very large, production greater than 3000 lb.
L — Large, production of 1000 to 3000 lb.
ML — Moderately large, production of 600 to 1000 lb.
M — Moderate, production of 300 to 600 lb.
S — Small, production less than 300 lb.

Large quantities of mica and other minerals were recovered from old fill, from pillars, and from thin skims of unmined pegmatite in some deposits; and recent operations in others were successful in part because of favorable market conditions and in part because of improvements in mining techniques over those previously used.

Methods of prospecting and early-stage development were much improved, especially in regions of thoroughly weathered rocks. Judicious search for pegmatite in areas of abundant float mica flakes or quartz blocks was accomplished by means of shallow trenching, deeper trenching with bulldozers or drag-line scrapers, and boring of test holes with hand augers. Such methods also were successful in many of the deeply weathered pegmatites of Brazil.[4]

The need for adequate surface exploration in advance of extensive underground development was repeatedly demonstrated, and it was further shown that the diamond drill can be an effective tool if its use is directed in close accord with known structural features of the pegmatite body being tested. Numerous blank holes bear testimony to the risks of overenthusiastic extrapolation from inadequate surface or subsurface data. Moreover, drill cores rarely provide satisfactory information on such features as the size and quality of mica books, coarseness of beryl, and general grade of the pegmatite. Nevertheless, the drill is valuable for locating extensions of known shoots or other lithologic units in pegmatites, or even for locating adjacent pegmatite bodies or shoots not elsewhere exposed and hence is useful in gathering data for rough calculations of reserves. In general, drill cores are most helpful if they are carefully interpreted in the light of structural and lithologic information from more complete exposures nearby.

Few pegmatites are completely exposed in three dimensions, so mineral shoots and incomplete zones may escape detection, either through lack of exposure or through removal by erosion, see Fig. 4. Commercial concentrations of pegmatite minerals rarely are coextensive with the pegmatite bodies that contain them, and hence they may lie undetected beneath the surface. In summarizing the structure of New England mica deposits, Bannerman and Cameron[1] state that the "pegmatites are mostly intrusive lenses, which . . . . occur . . . . at various levels with respect to the present erosion surface. Erosion has removed only the tops of some bodies, while others have been partly or almost entirely removed. If only the crest of a lens is exposed, whereas the minable mica-bearing zone is developed only along the flanks or keel, the pegmatite will be barren in surface exposures." Such relations undoubtedly are very common in most pegmatite districts,[4, 6, 9, 10] and hence hold considerable promise for future development of reserves of commercially desirable minerals.

Future operations in pegmatite districts might well be more efficient and less costly if conducted on the basis of extensive geologic data. Although there are few present criteria for recognition of workable concentrations not exposed at the surface, it commonly is possible to identify those pegmatites that offer little promise; hence they can be profitably eliminated from programs of exploration and development. The other deposits can be explored by means of trenches and test pits, and subsequently by diamond drilling and additional subsurface means if the results of the earlier work are sufficiently encouraging.

The history of mining in most pegmatite districts clearly indicates a correlation between activity and market conditions. Periods of great demand, high prices, and intensive mining are separated by longer periods of lower prices and small-scale, often sporadic, activity. The low production levels during these longer periods plainly are not caused by dwindling reserves or by suddenly increased difficulties in exposing additional shoots of commercial minerals. Thus, if market conditions appear to justify an increased production, attention might well be directed toward continuous operation of those mines known to yield the most return per unit of rock moved.

More efficient methods of mining, systematic recording of data on costs and yields for all parts of the mine,[4] the recovery of larger proportions of the mineral shoots, and the simultaneous recovery of more than one marketable product should greatly increase the chances for successful operations. More systematic and efficient mining might well be the result of a shift in attention from high-grade, pockety deposits to more extensive and continuous deposits of somewhat lower grade. The latter would lend themselves more readily to exploration in advance of mining, and efficient methods of mining and milling might lead to profitable operations. Careful mineralogic study of pegmatites and lithologic units within pegmatites is of great potential value, especially for recognition of recoverable byproducts. The possibilities for tantalum production from the Harding pegmatites, for example, might well have been recognized years ago if mineralogic studies had been carried on during the period of lepidolite mining. Studies of placer deposits, particularly those derived from pegmatites not previously worked, should yield information of possible value, so far as future mining of the heavier pegmatite minerals is concerned.

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Flotation Tests on Korean Scheelite Ore

by Will Mitchell, Jr., C. L. Sollenberger, and T. G. Kirkland

The beneficiation of a scheelite ore by flotation was investigated with emphasis on determining optimum conditions for obtaining maximum grade and recovery in the rougher concentrates. Variables such as pH, time, temperature, and pulp density during conditioning were plotted against the grade and recovery of the rougher concentrates. Several gangue regulators were tried, and several methods of cleaning rougher concentrates are described.

BENEFICIATION of a Korean scheelite ore has been studied during the past year in the Basic Industries Laboratory at Allis-Chalmers Manufacturing Co. A flowsheet has been recommended including flotation, gravity separation, and a final cleaning by a magnetic separator. The investigation was continued in the field of flotation in order to establish the effect of certain variables on recovery and grade of rougher concentrates. The results of this investigation are recorded here.

Mineralogy

The scheelite ore was obtained from Korea and was mined from the Sang Dong deposit. This deposit is described as lying in gently dipping metamorphosed sedimentary beds of Cambrian age.[1] The metamorphism was probably either metasomatic replacement or recrystallization, or both, with subsequent hydrothermal deposition. Igneous bodies in the form of granite porphyry have been found within 7 km of the main shaft and are considered to be a possible source of the mineralizing solutions. The Myobong formation, the host for the Sang Dong deposit, was laid down in middle Cambrian times and consists of series of mudstone, siltstone, shale, and marl, with local stratification of mica schists, hornfels, and tactites. The scheelite is found in quartz bands in the tactites. Although there are six of these bands, only one, the "main bed" averaging 1.75 pct WO_3, has been mined to any extent.

The ore sample as received in the laboratory was the product of a cone crusher and was all −2 in. The rock appeared to be even textured, medium to fine grained, and gray green in color. No scheelite could be recognized by the unaided eye. However, under ultraviolet light of 2537 A, the scheelite fluoresced and could be distinguished from the gangue. Under the black light, the scheelite showed no definite vein structure but was disseminated throughout the specimen.

The raw ore had a specific gravity of 2.96 and weighed about 125 lb per cu ft at −20-mesh Tyler. The −2-in. material was stage crushed through 20-mesh, and samples were cut for mineralogical, chemical, and spectrochemical analyses.

Several polished sections were cut from hand specimens. Figs. 1 and 2 show quartz to be the matrix for scheelite. It was reported by Kim[2] that the greatest scheelite concentration in the mine is found with a high concentration of quartz and a decrease in the abundance of garnet. Diameters of scheelite particles ranged from 1.2 mm down to less than 50 microns.

Spectrochemical analysis showed the ore to be high in aluminum, bismuth, calcium, iron, magnesium, manganese, silicon, tungsten, and with lesser amounts of beryllium, chromium, lithium, sodium, and titanium. Chemical assays showed the −20-

Will Mitchell, Jr., C. L. Sollenberger, and T. G. Kirkland, Members AIME, are Director, Ore Testing Engineer, and Mineralogist, respectively, of the Basic Industries Laboratory, Allis-Chalmers Mfg. Co., Milwaukee.

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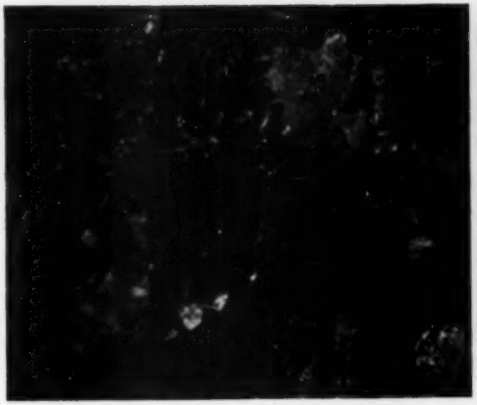

Fig. 1—Scheelite associated with quartz under reflected white light. (X35).

Fig. 2—Scheelite associated with quartz under reflected ultraviolet light. (X35).

mesh head to average 1.6 pct tungstate, 0.1 pct molybdate, and 0.2 pct bismuth. All tests for radioactivity were negative.

Bismuthinite and molybdenite also have been considered ore minerals, but only the flotation tests pertaining to scheelite have been included in this paper.

Table I presents a list of the significant minerals recognized in the feed, along with a rough estimate of the percentages by weight of the more abundant.

Particle counts on the individual screened fractions indicated that the scheelite was about 90 pct liberated in the −150 +200-mesh fraction. A few grains were liberated in the 20 to 28-mesh fraction. Eighty percent of the metallic minerals were liberated at 150-mesh.

The scheelite occurred as flat tabular grains, transparent and colorless. Particles fluoresced with three distinct color variations when exposed to black light. Isolation and chemical analyses of the three varieties proved that pure scheelite fluoresced blue, whereas that containing from 1 to 2 pct calcium molybdate fluoresced white, and that analyzing 4 pct or more calcium molybdate fluoresced yellow.[5] No segregation or selection of any particular variety in either the concentrates or the tails was noted when oleic acid was used as promoter.

An initial bulk sulphide float removed most of the opaque minerals; however, much of the pyrrhotite and the magnetite remained behind to be floated with the scheelite. In the sulphide float, bismuthinite appeared as elongated, sharp edged crystals, lead gray in color, and the molybdenite occurred as thin, foliated plates.

Minerals which reported in the scheelite concentrates are discussed later in the description of the floats.

Apparatus and Procedures

Flotation tests were conducted in a 500-g Fagergren test cell. The ore had been carefully stage crushed through 20-mesh and then split into approximately 500-g samples. Each sample was ground with 250 cc of distilled water in an Abbe 8x10-in. porcelain mill containing an 8-lb charge of flint pebbles. A 30-min grind was used at 58 rpm after which, the product analyzed 95.7 pct −150-mesh. The ground ore was transferred to the Pyrex glass cell and diluted to 25 pct solids with distilled water. The impeller of the test cell revolved at 1725 rpm.

Measurements of pH were made by means of a Beckman line-operated, replaceable glass electrode pH meter. Natural pH of the ore pulp in distilled water was 8.7 to 8.9.

In all tests a standard sulphide float was made to remove the sulphides, prior to scheelite flotation. The pulp was conditioned with reagent 301, 0.1 lb per ton, for 2 min; pine oil, 0.1 lb per ton, was added as a frother, and the sulphides were floated.

Numerous reagents have been proposed or used as collectors for scheelite. These include sodium oleate, Reagent 712, Reagent 708,[4] oleic acid, Reagent 825,[5] Orso,[6] laurylamine hydrochloride,[7] and Olate Flakes.[8] All of these reagents were investigated in this laboratory. Although slightly higher grade concentrates or higher recoveries were obtained with some of the above reagents than those obtained with oleic acid, the latter reagent was used in all the tests described in this report.

Table I. Mineralogy

Ore Minerals	Pct	Gangue Minerals	Pct	Gangue Minerals	Pct
Scheelite	1 to 2	Quartz	20 to 30	Magnetite	3
Bismuthinite	0.2	Biotite	10 to 20	Pyrrhotite	1
		Hornblende, Actinolite, Tremolite, Diopside, Hedenbergite	30 to 40	Pyrite	
				Chalcopyrite	1
				Wolframite	1
				Molybdenite	0.01
				Arsenopyrite	0.07
		Garnet	10	Galena	
		Fluorite	3	Ilmenite	
		Calcite	2		
		Apatite			
		Rutile			

Table II. Variation of Recovery and Grade with pH

pH	Weight Floated, Pct	WO_3 Grade, Pct	WO_3 Recovery, Pct
2.5	3.9	6.00	20.4
4.0	6.0	7.50	30.6
5.0	9.2	7.68	51.8
6.0	13.8	7.09	73.3
7.0	17.3	6.38	83.5
8.1	28.0	3.44	89.2
10.2	42.7	2.98	90.8
12.0	75.6	1.50	88.5

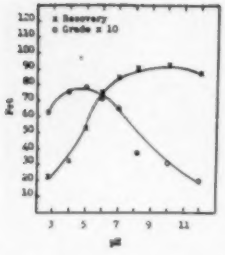

Fig. 3 (left)—Recovery and grade vs. pH. Conditioning at 26°C.

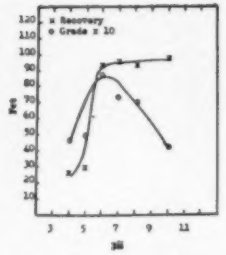

Fig. 4 (right)—Recovery and grade vs. pH. Conditioning at 58°C.

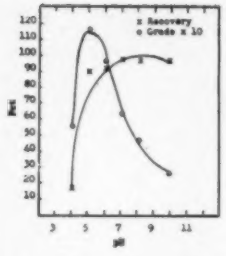

Fig. 5 (left)—Recovery and grade vs. pH. Conditioning at 80°C.

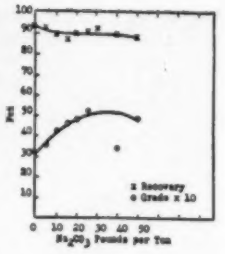

Fig. 6 (right)—Recovery and grade vs. Na_2CO_3 concentration.

To condition the pulp for several series of scheelite floats, pH or selectivity assisting reagents were added. Sufficient oleic acid was then added, (5 pct solution in wood alcohol) to obtain a concentration of 1 lb per ton. The pulp was conditioned in the test cell at 25 pct solids for 5 min. Air was then admitted and the froth collected until mineralization ceased, a period of about 5 min.

Chemical analyses of the low-grade dried products were made by the colorimetric method of Grimaldi and North.[9] High-grade concentrates were analyzed gravimetrically by a method described by Gleason.[10]

Variation of Recovery and Grade with pH

A series of floats was made in which the only variable was pH. It was varied from 3 to 12 by means of hydrochloric acid or sodium hydroxide. Conditioning was at room temperature, about 26°C. Results of these tests are listed in Table II.

These data, plotted on Fig. 3, show that maximum tungstate recovery was obtained at a pH of 10. Maximum grade was obtained at a pH of 5.

An examination of the concentrates under the petrographic microscope revealed that fluorite floated at each of the hydrogen ion concentrations investigated. At a pH of 4, gangue minerals floated were fluorite, magnetite and pyrrhotite. At pH 6, the concentrate also contained considerable biotite. At pH 10, the concentrate contained all of the foregoing gangue minerals, plus hornblende and pyroxenes.

pH Curves at Elevated Conditioning Temperatures

Since maximum recovery and maximum grade were obtained at different ends of the pH scale, it was decided to investigate the effect of elevated conditioning temperatures over the entire pH range.

These tests were made with the pulp temperatures raised to 58°C, and also to 80°C before the pH regulating agent and scheelite collector were added. The pulp was maintained at these temperatures during the 5-min conditioning period. The froth was then collected until mineralization ceased. Temperature drop during flotation was only 1 to 3 degrees. Tables III and IV show the results of these tests. Data are plotted on Figs. 4 and 5.

The configuration of the recovery and grade curves obtained at the higher conditioning temperatures was similar to that obtained at room temperature; however, the maximum points increased as conditioning temperature increased. It is interesting to note that maximum grade was obtained on Fig. 3 at a pH of 5, on Fig. 4 at a pH of 6, and on Fig. 5 again at a pH of 5. The maxima of the grade and recovery curves are widely separated on Fig. 3, almost coincide on Fig. 4, and are separated again on Fig. 5. The most advantageous conditioning temperature appears to be in the range of 58°C. One can conclude that conditioning temperature, as well as pH, is critical when floating scheelite from this ore.

Variation of Addition Agents

A series of floats was made in which sodium carbonate was added in amounts varying from 0 to 50 lb per ton. Conditioning was at room temperature. The results are summarized in Table V and are plotted on Fig. 6.

Additions of sodium carbonate up to 25 to 30 lb per ton yielded proportional increases in concentrate grade but recoveries remained practically constant. Additions greater than 30 lb per ton decreased concentrate grade and recovery.

Table III. Variation with pH at 58°C

pH	Weight Floated, Pct	WO_3 Grade, Pct	WO_3 Recovery, Pct
4	7.0	4.69	25.3
5	8.7	5.60	28.4
6	14.3	8.81	91.2
7	20.9	7.31	92.1
8	16.3	7.19	91.1
10	33.7	4.60	94.5

Table IV. Variation with pH at 80°C

pH	Weight Floated, Pct	WO_3 Grade, Pct	WO_3 Recovery, Pct
4	3.5	5.44	16.0
5	11.7	11.6	87.7
6	12.1	9.38	89.3
7	27.7	6.19	95.1
8	27.5	4.50	94.4
10	48.0	2.44	94.1

Table V. Sodium Carbonate

NA_2CO_3 Lb per Ton	pH	Weight Floated, Pct	WO_3 Grade, Pct	WO_3 Recovery, Pct
0	8.6	38.7	3.13	92.4
5	10.4	27.5	3.56	91.7
10	10.6	21.6	4.25	88.6
15	10.6	20.5	4.50	87.7
20	10.7	28.3	4.60	88.8
25	10.8	26.4	5.03	89.4
30	10.8	24.2	5.00	89.8
40	10.9	31.2	3.38	88.5
50	11.1	25.6	4.75	88.2

The previous tests with NaOH had shown a decrease in concentrate grade with increased pH. With Na_2CO_3, however, an increase in grade was noted as pH increased. Petrographic comparisons of concentrates obtained at pH 10 with each reagent revealed that the concentrate obtained using NaOH contained more mica and fine gangue than the concentrate obtained using Na_2CO_3. Sodium carbonate was apparently a depressant for mica and appeared to be a better dispersant of gangue minerals than sodium hydroxide.

Sodium silicate additions were varied from 0 to 9 lb per ton, while the pH was maintained at 10 with sodium hydroxide. Conditioning was at room temperature. The results obtained are listed in Table VI and are plotted on Fig. 7.

These data show that concentrate grade increased with increased concentrations of sodium silicate, whereas recovery decreased. Petrographic examinations revealed that 5 lb per ton sodium silicate depressed the quartz and the green silicate minerals, and concentrations of 7 to 9 lb per ton also depressed much of the biotite.

Three successive floats from a pulp containing 8 lb per ton sodium silicate were made adding 1 lb per ton oleic acid for each. It was found that 48 pct of the scheelite was recovered in the first float at a 24 pct grade, 30 pct in the second at a 14 pct grade, and 7 pct in the third at a 3.8 pct grade. This made a total recovery of 85 pct at an average grade of 14 pct WO_3. Such a scheme has the advantage of producing a relatively high-grade concentrate immediately but would probably not be desirable since it requires three stages of flotation from the bulk of the feed.

A series in which varying quantities of hydrofluoric acid were added showed that as the pH was reduced, both recovery and grade decreased until all minerals were depressed at a pH of 5.3.

A few additional rougher tests showed that a slightly increased recovery of WO_3 could be obtained if a frother having high wetting properties, such as Pentasol 124, was used instead of pine oil. A slightly increased rougher concentrate grade at essentially the same recovery could be obtained by spraying the froth with about 0.4 lb per ton Aerosol OT in the form of a 1 pct water solution.

Variation of Pulp Density During Conditioning

The effect of pulp density during conditioning was investigated in the following manner. After the sulphides were floated by the previously described method, the pulp was placed in a 2-liter beaker and decanted in successive tests to 35, 50, and 64 pct solids. To each pulp was added 1.9 lb per ton Na_2SiO_3, 1.0 lb per ton Na_2CO_3, and 1.0 lb per ton oleic acid. The pH was maintained at 10. Pulps were conditioned at room temperature for 5 min in the beaker by agitating with the impeller of the Fagergren test cell. Pentasol 124, 0.2 lb per ton, was added as a frother. The material was transferred to the test cell, diluted to 25 pct solids and floated. The results obtained are compared on Table VII and on Fig. 8.

These data show that grade increased and recovery decreased slightly with increased pulp density during the conditioning period. These effects were most pronounced when the pulp was conditioned at 64 pct solids.

Table VI. Sodium Silicate

Na_2SiO_3 Added Lb per Ton	pH	Weight Floated, Pct	WO_3 Grade, Pct	WO_3 Recovery, Pct
0.0	10	43.7	2.98	90.6
0.2	10	31.3	4.43	90.6
0.4	10	28.5	4.53	88.0
0.8	10	20.6	6.33	88.6
1.9	10	11.6	9.88	83.4
3.9	10	7.2	12.6	77.0
5.0	10	6.6	12.3	79.4
7.0	10	3.0	23.2	39.7
9.0	10	2.2	28.6	48.4

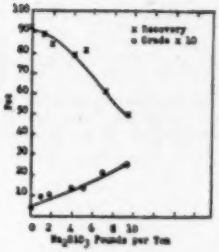

Fig. 7 (left)—Recovery and grade vs. Na_2SiO_3 concentration at pH 10.

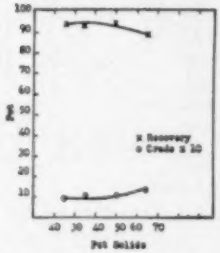

Fig. 8 (right)—Recovery and grade vs. pulp density during conditioning.

Effect of Conditioning Time

From the data of the foregoing tests, it appeared that conditioning at a pulp density of 50 pct solids increased the concentrate grade with a minor decrease in recovery. A series of tests was made with pulp density during conditioning maintained at 50 pct solids, but with the conditioning time varied from 5 to 40 min. The same reagent combination was used as in the pulp density series, and conditioning was at room temperature. The results obtained are tabulated below and are plotted on Fig. 9.

A decrease in concentrate grade with increased conditioning time was recorded. Recovery remained relatively constant.

Cleaning the Rougher Concentrate

Cleaning the rougher concentrate by flotation was found to be extremely difficult because of the tenacious reagent coating on both scheelite and gangue minerals. A cleaner float without additional re-

Table VII. Variation of Pulp Density During Conditioning

Pulp Density Pct Solids	Weight Floated, Pct	WO_3 Grade, Pct	WO_3 Recovery, Pct
25	14.2	8.94	93.3
35	16.3	9.44	92.3
50	16.3	9.69	93.0
64	11.6	12.2	89.0

Table VIII. Effect of Conditioning Time

Conditioning Time, Min	Weight Floated, Pct	WO_3 Grade, Pct	WO_3 Recovery, Pct
5	16.3	9.69	93.0
10	19.6	7.75	92.9
20	22.7	6.94	93.6
40	25.7	5.86	93.8

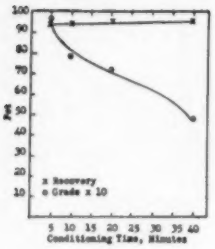

Fig. 9—Recovery and grade vs. conditioning time, conditioning at 50 pct solids.

agents increased the grade to 17.1 pct. Recovery in this concentrate dropped to 81.9 pct. However, with the addition of sodium silicate to the rougher concentrate followed by two cleaning operations, concentrate grade was increased to 27 pct. Recovery in the final concentrate without recirculation of middlings dropped to 68 pct.

A sample of rougher concentrate was roasted for 1 hr at 340°C, to remove the oleic acid coatings. By a series of cleaner floats on the roasted concentrate using sodium silicate as the regulator and about 0.6 lb per ton oleic acid in stages, a concentrate assaying 27.7 pct WO_3 was again obtained which contained 71.2 pct of the total WO_3 in the raw ore without recirculation of middlings. The concentrate in both cases contained only fluorite, scheelite and traces of magnetite and pyrrhotite.

It was found that the scheelite, after the cleaning operations, could be depressed with quebracho and the fluorite then floated with oleic acid. The fluorite float assayed only 5 pct WO_3. The nonfloat assayed 40 pct WO_3 and contained 61 pct of the WO_3 in the original ore. This was the highest concentrate grade attained by means of the flotation process.

At present the laboratory is continuing research on several reagents that appear to be much more selective to scheelite than is oleic acid.

Summary

An effort was made to show the effect of the variables usually encountered in the flotation of nonmetallic minerals on recovery and grade of rougher flotation concentrates from a complex scheelite ore, when using oleic acid as the scheelite promoter. The variables investigated were pH, conditioning time, pulp density during conditioning, conditioning temperature, and quantity of several gangue regulators.

In this investigation, the most interesting results, from an academic viewpoint, were those obtained when pH was varied at three temperature levels of conditioning. At room temperature, the peaks of the grade and recovery curves were five pH units apart. However, at 58°C, the peaks of recovery and grade percentage curves were both higher, and occurred at practically the same pH. At 80°C, still higher maximums were obtained, but the maximum points were about 2 pH units apart.

Concentrate grade equivalent to that obtained at elevated temperatures but at a slightly lower recovery was obtained with room temperature conditioning by using only 2 lb per ton sodium silicate as a gangue depressant. This reagent was the most effective gangue regulator investigated.

An increase in pulp density during conditioning caused a slight increase in grade and a slight decrease in recovery. Conditioning periods greater than 5 min caused rapid decreases in grade but had little effect upon recovery.

Rougher concentrates were cleaned to 27 pct WO_3, using additional sodium silicate as a regulator in the cleaner circuits. The cleaned concentrate was upgraded to 40 pct WO_3 by depressing the scheelite with quebracho and then floating the fluorite gangue from the scheelite.

Conclusions

From the discussion and the data recorded in this report it can be concluded that careful temperature control must be maintained during elevated temperature conditioning in a scheelite oleic acid system. Higher than room temperatures, up to a certain point, improve recovery and grade; however, beyond this point advantage is reduced. It has been proved in the tests that the pH required for optimum results varies according to the temperature of conditioning.

From an economic standpoint, elevated temperature conditioning does not appear to be practical, because equivalent grade at only slightly decreased recovery can be obtained by conditioning at room temperature and by using moderate amounts of sodium silicate as a gangue depressant.

Conditioning at high pulp densities or for periods greater than 5 min does not seem to offer any advantages. In fact, extended conditioning periods have proved deleterious to grade of concentrate.

The most satisfactory reagent combination used was found to be 1 lb per ton oleic acid, 2 lb per ton sodium silicate, 1 lb per ton sodium carbonate, 1/10 lb per ton pine oil, and 1/10 lb per ton wetting agent. A pH of 10 should be maintained if the separation is carried on at room temperature.

Acknowledgments

The authors wish to acknowledge with thanks Anne M. Cullen for her assistance in conducting many of the tests, also Arneth J. Calton for directing the analytical work. Thanks are also due to Herbert Kim and to Arthur S. Hecht for giving us the benefit of their wealth of experience in the field of scheelite concentration.

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aime NEWS

Don't Miss The Annual Meeting

WHERE: St. Louis

WHEN: Feb. 19-22

WHAT: Six Divisions holding 79 sessions plus social events, field trips, and business meetings.

What are you interested in? Taxation, zinc, sulphur, depreciation, titanium, clay? Or is it geology, crushing, ground water, Canadian minerals, coal, or the ECA? These and all the other topics of interest to miners, mill men, metallurgists, geologists and petroleum men will be discussed at 79 sessions during the Annual Meeting in St. Louis. Those who have attended past meetings will need no coaxing this time. Members who have never been to an Annual Meeting can scan the list of mining papers and the meeting program printed here, and imagine what they'll be missing if they're not at the registration desk on Feb. 19.

The Council of Section Delegates will be in St. Louis early on Feb. 17 for an all-day meeting. On Sunday, the 18, the Board of Directors will meet, the MBD will hold a business session, and the MIED's educators will hold afternoon and evening gatherings at Washington University.

Technical sessions will begin on Monday morning, and the first big social item, the Welcoming Luncheon, is set for 12:15 on that day. Arthur Holly Compton, noted atomic scientist and chancellor of Washington University, will be the speaker at that affair. Dr. Compton, a veteran of the Manhattan Project, the atom bomb project, is also vice-chairman of the U.S. National Commission for UNESCO. High honors have been bestowed on him by learned societies and governments the world over.

Evening relaxation from Monday's 23 sessions will be provided at the all-male Dinner-Smoker. Tuesday morning finds presiding officers and speakers for the day in a huddle over the 23 more sessions to be held between 9 am and 5 pm. Luncheons that day will include those of the MIED and the Mining, Geology and Geophysics Div. In the evening, the Metals Branch will gather for dinner, as will the Petroleum group, and the Society of Economic Geologists. The Jefferson Hotel will be the scene of the perennial Informal Dance.

Hardy MBD members will rise early on Wednesday and listen for the music of bagpipes at their traditional Scotch Breakfast. Highlights for the day will include the IMD lecture, and that Division's Annual Business Meeting. The Coal, Industrial Minerals, and the Mineral Economics Divisions will regale themselves at Wednesday luncheons.

The Annual Banquet is scheduled for Wednesday, and incoming AIME President W. M. Peirce will be on hand to address the gathering.

The final day, Thursday, will be a light one, with only nine sessions. The Minerals Beneficiation Div. will hold their luncheon on that day. The final session at the meeting, but one of the most important, will be the Careers Forum for students, to be held on the campus of the University of Missouri with Dean Curtis L. Wilson presiding. A panel of top-flight engineers will be on hand to answer questions from the students and discuss career problems.

WOMEN'S AUXILIARY PROGRAM

The WAAIME, under the general chairmanship of Mrs. Howard I. Young, will be active for all four days of the meeting. On Monday, Feb. 19, they will attend the Welcoming Luncheon, and that evening will hold a dinner and show at the Starlight Roof of St. Louis' Chase Hotel. Tuesday morning will find the ladies gathering for their own Annual Meeting at the Jefferson, and later at the Statler Roof for a luncheon and fashion show. Wednesday morning's feature will be a round table at the Jefferson, presided over by Mrs. Curtis Wilson. On the final day of the meeting, a sightseeing tour will be conducted, courtesy of the St. Louis Section, with a luncheon en route.

FIELD TRIPS

More detailed data on Annual Meeting field trips has been made available since our December issue. Get your reservations in early for a place on these interesting forays.

Trips 1-5 Friday, Feb. 23

No. 1 Zinc smelters, Monsanto and Fairmont, Ill. Leave Jefferson Hotel 9:00 am and return 5:00 pm. Price $3.50 (includes lunch).

No. 2 Disseminated lead belt, Bonne Terre and Herculaneum, Mo. Leave Jefferson Hotel 8:00 am and return 5:00 pm. Price $6 (includes lunch).
Mills and smelter, St. Joseph Lead Co. (Restricted to 150 persons, no ladies allowed.)

No. 3 Coal mining and washing plant, Millstadt, Ill., and coking plant of Koppers Co., Granite City, Ill. Leave Jefferson Hotel 9:00 am and return 4:30 pm. Price $3.50 (includes lunch).

No. 4 Pittsburgh Plate Glass Co., mine and plant, Crystal City, Mo. Leave Jefferson Hotel 9:00 am and return 4:30 pm. Price $3.50 (includes lunch).

No. 5 U. S. Bureau of Mines, synthetic oil plant, Louisiana, Mo. Leave Jefferson Hotel 8:00 am and return 5:00 pm. Price $6 (includes lunch).

No. 6 Tri-State Zinc District, Missouri, Kansas, and Oklahoma. Leave St. Louis 11:20 pm Thursday, Feb. 22, arrive in Joplin 8:30 am Feb. 23. Leave Joplin 9:00 pm Feb. 23 and return to St. Louis 7:45 am Feb. 24. Approximate cost $35 depending upon Pullman accommodations. To provide adequate Pullman space, reservations for this trip must be made before Feb. 1.

Thursday, Feb. 22

A Lead mines and mills at Bonne Terre, Mo. Leave Jefferson Hotel 8:00 am and return 5:00 pm. Price $6 (includes lunch). (Restricted to 150 persons. Ladies will not be permitted on this trip.)

Technical Program

INDUSTRIAL MINERALS DIVISION

MONDAY, A.M. and P.M.

Canadian Industrial Minerals

Canada's Wealth in Industrial Minerals. M. F. Goudge, chief, Canada Dept. of Mines & Technical Surveys, Ottawa, Canada.

Geology and Mining and Milling Practices at St. Lawrence Fluorspar Deposits, St. Lawrence, Newfoundland. Claude K. Howse, government geologist, Newfoundland Dept. of Natural Resources.

Industrial Minerals of Quebec. P. E. Bourret, Quebec Dept. of Mines.

Asbestos Mining at the Jeffrey Mine. K. V. Lindell, mine manager, Canadian Johns-Manville Co., Ltd.

Saskatchewan's Industrial Minerals. A. J. Williams, Saskatchewan Dept. of Natural Resources.

Saskatchewan Minerals, Sodium Sulphate. G. F. Miller, Saskatchewan Minerals Co.

Industrial Minerals in British Columbia and Their Relations to Power, Fuels, and Markets. T. E. Hartley Sargent, chief mining engineer, British Columbia Dept. of Mines.

Windemere Gypsum Deposits, Rocky Mountains, British Columbia. J. W. McCammon.

MONDAY, P.M., WEDNESDAY, A.M. and P.M.

Clay Morphology, joint session with Geology Subdivision and SEG.

Paper listed under Geology Subdivision.

MONDAY, P.M.; WEDNESDAY, A.M.; and THURSDAY, A.M.

General Sessions to be filled by the following papers:

Synthetic Mica Research at the Colorado School of Mines. William C. Aitkenhead, State College of Washington.

Elemental Phosphorous and the Commercial Importance of Molecularly Dehydrated Phosphates. J. G. Miller, Westvaco Chemical Co.

Beryl. (A general paper) Gordon F. Simons, Beryllium Corp.

Volcan Overo Sulphur Mine, Argentina. B. R. Burnett, general manager, Sominar Sociedad Minera Argentina; and Howard A. Meyerhoff, administrative secretary, American Assn. for the Advancement of Science.

Factors Influencing the Localization of Fluorspar in Southwestern Grant Co., New Mexico. Elliot Gillerman, U. S. Geological Survey, New Mex.

Occurrence and Properties of Southern Illinois Silica. John E. Lamar, Illinois Geological Survey.

Fluorspar Deposits of Mexico. Robert M. Grogan, Illinois Geological Survey.

Origin of Emery Ore Deposits. Gerald M. Friedman, University of Cincinnati.

Searles Lake Operation at Trona, Calif. J. E. Ryan.

Shale Planer. Clare B. Laird, Eagle Iron Works.

Ruberoids New Lowell Asbestos Mine and Mill. Michael J. Messel, general superintendent, Vermont Asbestos Mines Div., Ruberoid Co.

TUESDAY, A.M.

Dimension Stone

Some Economic Aspects of the Southern California "Black Granites". R. H. Hoppin and L. A. Norman, Jr.

Operational Studies in the Pennsylvania Slate Industries. W. F. Mullin, Pennsylvania State College.

A Suggested Method of Quarrying a Boulder-Type Granite Formation. W. P. Mould, director of research, Rock of Ages Corp.

Morton Granite Gneiss of the Minnesota Valley. E. H. Lund, University of Minnesota.

Dimension Stone in Minnesota. G. A. Thiel and G. M. Schwartz, director, mining geological survey, University of Minnesota.

Building Stones of Indiana. John B. Patton, Indiana Geological Survey.

TUESDAY, A.M.

Clay Technology

Ball Clays: Control Testing Applied to Exploration and Mining. G. W. Phelps, G. W. Lee, and G. G. Brun, United Clay Mines Corp.

Origin, Processing and Application in the Trades of "Florida Phosphate Slimes". Poole Maynard, industrial geologist, Atlantic Coast Line Railroad.

Grinding to Sub-Sieve Ranges with Jet Pulverizers. J. B. Chatelain and M. F. DuFour, Micronizer Co.

Ceramic Investigation of an Arkansas Clay Deposit. W. J. Smothers and Theodore Dziemianowicz, University of Arkansas.

TUESDAY, P.M.

Light-weight Aggregates

Perlite in Oregon, Washington, and Idaho. K. E. Hamblen, consulting engineer, Portland, Ore.

A Summary Statement of Perlite Trends. Clarence R. King, consultant.

Lightweight Aggregate from Kansas Clays. Norman Plummer, Kansas Geological Survey.

Economics of Lightweight Aggregate in the United States. Staff, Battelle Memorial Institute.

TUESDAY, P.M.

Drilling and Pipe Transportation. Joint Session with Mining Subdivision.

For list of papers see Mining Subdivision program.

WEDNESDAY, P.M.

Cost Accounting in Mining. Joint Session with MED.

For list of papers see MED program.

THURSDAY, A.M.

Economics of Industrial Minerals. Joint session with MED.

For list of papers see MED program.

WEDNESDAY, Noon

Luncheon.

COAL DIVISION

MONDAY, A.M.

Coal Depreciation Problem

Co-chairmen: J. W. Woomer, J. W. Woomer & Associates; and L. C. McCabe, chief, Office of Air & Stream Pollution Research, U. S. Bureau of Mines

Authors to include: (1) R. M. Kelday, Steel Co. of Canada, Ltd., (2) D. L. McElroy and George W. Kratz, Pittsburgh Consolidation Coal Co., (3) Lee Barrett and E. A. Reilly, Joy Mfg. Co.

MONDAY, P.M.

Coal Depreciation Problem

Co-chairmen: Clayton G. Ball, vice-president, Paul Weir Co.; and Elmer R. Kaiser, assistant director of research, Bituminous Coal Research, Inc.

Authors to include: (1) R. H. Swallow and Roy Dean, Ayrshire Collieries Corp. (2) T. H. Bierce, Rochester & Pittsburgh Coal Co., (3) Thomas Cheasley and Roy Park, Sinclair Coal Co.

TUESDAY, A.M.

General Session

Co-chairmen: E. R. Price, manager, coal properties, Inland Steel Co., Wheelwright, Ky.; and J. P. Blair.

Economic Significance of Recent Technological Research on Solid Fuels. A. C. Fieldner, U. S. Bureau of Mines.

The Frothing Characteristics of Cresylic Acids. Shio-Chuan Sun, assistant professor of mineral preparation, Pennsylvania State College.

Operating Data for the Colmol. Clifford Snyder, Sunnyhill Coal Co.

TUESDAY, P.M.

General Session

Co-chairmen: W. A. Mueller, professor of mineral preparation, Ohio State University; and B. W. Gandrud, preparation engineer, U. S. Bureau of Mines.

Effect of Oxidation on the Flotability of Coals. Shio-Chuan Sun, assistant professor of mineral preparation, Pennsylvania State College.

Efficiency and Sharpness of Separation in Evaluating Coal-Washery Performance. H. F. Yancey, supervising engineer, U. S. Bureau of Mines; and M. R. Geer, U. S. Bureau of Mines, Seattle, Wash.

Characteristics of Mechanized Mining Sections. Arnold W. Asman, chief, div. of mining; and A. W. Bitner, Pennsylvania State College.

WEDNESDAY, A.M.

Preparation

Co-chairmen: Byron M. Bird, technical consultant, Jeffrey Mfg. Co.; and Charles C. Wright, vice-president, Oilwell Research, Inc.

An Approximate Method for Predicting and Comparing the Results to be Expected When Dewatering Coal by Means of Centrifuges. Orville R. Lyons, manager of preparation, Republic Steel Corp.

The Biological Formation of Sulphur and Acid Mine Water. K. L. Temple, A. R. Colmer, and W. A. Koehler, West Virginia University Engineering Experiment Station.

Closed Water Circuits with Cyclone Thickeners—Fact or Fiction. Victor Phillips, Heyl & Patterson.

WEDNESDAY, A.M.

Geology. Joint session with SEG

Co-chairmen: Gilbert H. Cady, head, coal div., Illinois Geological Survey; Paul H. Price, state geologist, W. Va. Geological Economic Survey.

A Review of Recent Published Studies on Coal Geology, Exclusive of Research Studies. A. T. Cross, West Virginia Geological Survey.

A symposium on current academic and laboratory activities in the Middle West. Participants to include: J. Norman Payne, University of Arkansas; Arthur Bevan, Illinois Geological Survey; Charles Wier, Indiana Geological Survey; Garland Hershey, Iowa Geological Survey; Walter Shoewe, University of Kansas; C. S. Crouse, University of Kentucky; W. V. Searight, Missouri Geological Survey; Robert H. Dott, Oklahoma Geological Survey; James M. Schopf, U. S. Geological Survey; and Byran C. Parks, U. S. Bureau of Mines

WEDNESDAY, Noon

Luncheon

WEDNESDAY, P.M.

Geophysics. Joint Session with SEG

Co-chairmen: H. F. Yancey, supervising engineer, U. S. Bureau of Mines, Seattle; Clayton G. Ball, vice-president, Paul Weir Co.

Coal Resources Studies in the United States. Paul Averitt, U. S. Geological Survey.

An Oxidation Method for Investigating the Petrographic Composition of Some Coals. R. Q. Shotts, associate professor, School of Mines, University of Alabama.

Cyclic Sediments and Engineering Geology. S. S. Philbrick, U. S. Army Engineers.

Some Oxidation Effects on Coal Mine Roofs. Charles T. Holland, professor and head, mining engineering dept., Virginia Polytechnic Institute.

Differential Thermal Curves of Selected Arkansas Coals. G. M. Gamel, Jr. and W. J. Smothers, University of Arkansas.

MINERALS BENEFICIATION DIVISION

MONDAY, P.M.

Materials Handling and Solid-Fluid Separation

TUESDAY, A.M.

Symposium: What's New in Milling Equipment

There will be eight or more speakers describing the new equipment in milling.

TUESDAY, P.M.

Concentration

WEDNESDAY, A.M.

Scotch Breakfast

Entertainment, of course; and the scrambled eggs can be selected "with" or "without".

WEDNESDAY, A.M.

Crushing and Grinding, and Operating Control in Milling Operations

THURSDAY, A.M.

Joint session with the Extractive Metallurgy Div.

THURSDAY, Noon

MBD Luncheon

THURSDAY, P.M.

Clean-up session for papers and discussions carried over from other sessions.

MINERAL ECONOMICS DIVISION

MONDAY, P.M.

Foreign Minerals

Foreign Mineral Developments (Except ECA Areas) Elmer W. Pehrson, acting regional director, Foreign Region IX, U. S. Bureau of Mines.

ECA Activity in Stimulating Foreign Mineral Production. Charles E. Stott, ECA, Washington, D. C.

Mineral Position of the USSR. Paul M. Tyler, mineral technologist and economist, Bethesda, Md.

Principal Mineral Strategic Problems in a Conflict Between the East and West. Author to be announced.

TUESDAY, A.M.

Exploration Economics. Joint session with Mining, Geology, and Geophysics Div.

Economics of Geophysics. Sherwin F. Kelly, president, Sherwin F. Kelly Geophysical Services.

Cost of Geophysical Exploration. Carl A. Bays, Carl A. Bays and Associates.

A Graphite Statistical History of the Tri-State District. John S. Brown, chief geologist, St. Joseph Lead Co.

Additional papers to be announced.

TUESDAY, Noon

Luncheon followed by short business meeting and forum discussion of the AIME "controversial matters" resolution.

TUESDAY, P.M.

Taxation, Tariffs, and Subsidies

An Operator Looks at Mine Taxation. Louis Ware, president, International Minerals & Chemical Corp.

The Depletion Allowance—Loophole or Economic Necessity. Maurice Peloubet, Pogson, Peloubet & Co.

A Legislative Income Tax Program for Mineral Producers. Granville S. Borden, special tax counsel, Standard Oil Co. of California.

Free Markets, Tariffs, and Subsidies. Felix E. Wormser, vice-president, St. Joseph Lead Co.

The Case for Mine Subsidies. O. W. Bilharz, president and general manager, Bilharz Mining Co.

WEDNESDAY, P.M.

Cost Accounting in Mining. Joint session with Industrial Minerals Div.

Consideration of Economic Factors in Establishing Fair Value of Mineral Properties. Bleeker L. Wheeler, consulting engineer.

Henry Fayol Method of Administrative Organization. Pierre deVitry, Patented Marble & Granite Machinery Co.; and W. P. Mould, director of research, Rock of Ages Corp.

Engineers and Accountants—A Partnership. Edward Tubb, comptroller, International Minerals & Chemical Corp.

Additional papers to be announced.

THURSDAY, A.M.

Economics of Industrial Minerals. Joint session with Industrial Minerals Div.

Economic Trends of Fluxstone—Middle Atlantic Area. John A. Ames, geologist, Baltimore & Ohio Railroad Co.

California's Mineral Position in a Changing Industrial World. Olaf P. Jenkins, chief, California Div. of Mines.

Mineral Leasing on Government Acquired Lands. J. D. Turner, U. S. Geological Survey, McAlester, Okla.

Economics of the Potash Industry. Author to be announced.

Economics of Future Sulphur Supplies. Author to be announced.

MINERAL INDUSTRY EDUCATION DIVISION

SUNDAY, P.M.

Graduate Programs in the Field of Mineral Engineering.

Graduate Programs and Research in Petroleum Engineering. Harry H. Power, professor and chairman, petroleum engineering dept., University of Texas.

Lehigh Graduate School of Metallurgical Engineering Practice. R. D. Stout, Lehigh University.

Graduate Courses in Mining Engineering. M. D. Cooper, director, mining engineering education, National Coal Assn.

Research in Mining Engineering. Eugene P. Pfleider, associate professor of mining, University of Minnesota.

Discussion: John Chipman, Massachusetts Institute of Technology; and Harold L. Walker, professor of metallurgical engineering, University of Illinois.

SUNDAY, P.M.

Buffet supper

SUNDAY, Evening

Graduate Engineering Training. S. C. Hollister, dean of engineering, Cornell University.

MONDAY, A.M.

Teaching of Metallurgical Engineering. John L. Bray, professor, metallurgical engineering, Purdue University.

Vocational Opportunities for Geology Graduates in the Field of Industrial Rocks and Minerals. Walter D. Keller, professor of geology, University of Missouri.

Enrollment—Mineral Industry Field. W. B. Plank, Lafayette College.

Reports from Committee Chairmen.

TUESDAY, Noon

Luncheon

MINING, GEOLOGY, AND GEOPHYSICS DIVISION

Preprints of Mining, Geology, and Geophysics Div. Papers Available

Outlines of talks scheduled by the Mining, Geology, and Geophysics Div., at the St. Louis Annual Meeting, February 1951, are being mimeographed for immediate free distribution on request. The speakers on the MGGD program are submitting summaries up to 2000 words long to give AIME members plenty of opportunity to prepare discussions in advance of the meeting. This procedure has been inaugurated by MGGD as a convenience to members and to foster increased discussion at the sessions.

The papers listed below are those for which it is expected that mimeographed outlines will be available. This is not the final program as there will be changes and additions.

You are invited to list the numbers of the titles that interest you, sign your name and address in the box below, and tear out and mail as directed. The mimeographed summaries will be sent to you promptly. In return, formal contributions in the form of prepared discussions will be welcomed by the men listed as "organizers" of the various sessions. Their addresses can be found in the AIME directory.

A word of caution: These preprints are, in some cases, rough drafts. Authors expect to revise them. They do not constitute publication, and publication rights are reserved.

MINING SUBDIVISION

General Session organized by E. D. Gardner, U. S. Bureau of Mines.

(1) *Illinois Operations of the Eagle Picher Mining & Smelting Co.* Claude O. Dale, Eagle-Picher Mining & Smelting Co.; and W. J. Rundle, associate professor, mining engineering, University of Wisconsin.

(2) *Progress and Trends in Mining Practices on the Menominee Range, Michigan.* Philip D. Pearson, Menominee range superintendent, Inland Steel Co.

Drilling and Pipe Transport. Joint session with Industrial Mining Div. Organized by E. D. Gardner.

(3) *Pipe Line Transportation of Phosphate.* I. S. Tillotson, R. B. Burt, James A. Barr, Jr., research engineering, chemical div., Armour Co.

(4) *Tungsten Carbide Drilling on the Marquette Range.* Eugene Lillstrom.

General Session organized by E. D. Gardner.

(5) *Dragline Stripping.* R. H. Whitney and C. E. McManus, assistant general manager, Minnesota mines, M. A. Hanna Co.

(6) *Planning, Development, and Operations at Embarass Lake Mine.* James R. Stuart.

(7) *Factors Influencing Choice of Loading Machine.* Donald W. Mitchell.

(8) *Underground Electrical Installations at Miami Copper Co.* B. R. Coil, assistant general manager, Miami Copper Co.

General Session organized by E. D. Gardner.

(9) *On-the-Job Training of Mine Employees at Utah Copper Mine.* Howard B. Gundersen.

(10) *Crystallographic Orientation of the Diamonds.* Eugene P. Pfleider, associate professor of mining, University of Minnesota.

(11) *An Unusual Test of the Accuracy of Well Surveying.* S. H. Williston, vice-president, Cordero Mining Co.

GEOLOGY SUBDIVISION

Diaspore and Other Hydroxides. Field and Laboratory Studies. Joint session with SEG and Ind. Min. Div. Organized by A. F. Frederickson, associate professor of geology, Washington University, St. Louis.

(12) *Genetic Significance of the Aluminum Hydroxide Minerals.* A. F. Frederickson.

(13) *Genesis of Certain Clay Minerals.* E. Hauser.

(14) *New Clay Mineral Evidence Concerning the Diagenesis of Some Missouri Fire Clays.* J. F. Burst.

(15) *Structure of the Aluminum Oxides and Hydroxides.* W. O. Milligan and J. L. McAtee.

(16) *Studies in the System Al_2O_3-SiO_2-H_2O.* R. Roy and E. F. Osborne.

(17) *Distribution and Occurrence of the Mercer Fire Clay, North-central Pennsylvania.* R. C. Bolger and J. H. Weitz, associate professor of geology, Lehigh University.

(18) *Random Observations on the Origin of Missouri Diaspore.* Walter D. Keller, professor of geology, University of Missouri.

(*Continued on next page*)

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(Mining, Geology, Geophysics Papers Only)

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Papers and Program Committee
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Room 4203, GSA Building
Washington 25, D. C.

(19) *Kisameet Bay Clay Deposit.* E. Hauser.

(20) *Examples of Bauxite Deposits Illustrating Variations in Origin.* E. C. Harder, Aluminum Secretariat Ltd., Montreal, Canada.

(21) *Composition of Argillaceous Rocks in Relation to Conditions of Genesis.* G. Millot.

(22) *Changes in Clay Mineral Character with Distance of Transportation.* R. A. Rowland.

(23) *Interrelationships Between Structure and Genesis in the Kaolinite Group.* T. F. Bates.

(24) *Genesis and Morphology of the Alumina-Rich Laterite Clays.* G. D. Sherman.

(25) *Alteration of Phosphatic Limestone to Montmorillonite Clay.* F. R. Hunter, research geologist, International Minerals & Chemical Corp.

Geology and Choice of Mining Methods. Joint session with Mining Subdivision. Organized by George B. Clark, assistant professor of mining engineering, University of Illinois.

(26) *Design of Underground Mines.* R. S. Loofbourow, manager, mining div., E. J. Longyear Co.

(27) *Structural Geology as the Setting for Mining Operations.* Edward H. Wisser, consulting mining engineer, San Francisco.

(28) *Application of Geology to Mining at Giant Yellowknife.* J. D. Bateman, chief geologist, Giant Yellowknife Gold Mines, Ltd.

(29) *Relation of Geology to Mining Methods at Climax.* M. S. Walker.

Symposium on Ground Water Geology. Joint session with SEG, organized by George W. White, professor of geology, University of Illinois (SEG).

Papers not available for preprinting.

Surface and Near-surface Manifestations of Ore Bodies. Joint session with SEG organized by Roger H. McConnel, chief geologist, Bunker Hill & Sullivan Mining Co.

(30) *Surface Expressions of Veins in the Pachuca Silver District, Mexico.* C. L. Thornburg, geologist, U. S. Smelting, Refining & Mining Co.

(31) *Ore Shoots of the Bolivian Tin Deposits.* F. S. Turneaure.

(32) *Abnormal Copper, Lead, and Zinc Contents of Soil Near Metalliferous Veins.* Lyman Huff.

(33) *Guides to Ore in the Pachuca Silver District, Mexico.* Edward H. Wisser consulting mining geologist, San Francisco.

(34) *Shallow Expressions at Silver Belt Ore Shoots, Coeur d'Alene District, Idaho.* Robert E. Sorenson, chief engineer and geologist, Hecla Mining Co.

(35) *The Fairview Mine, Similkameen District, B. C.* C. O. Swanson.

(36) *Outcrops in Limestones as Ore Guides.* Charles H. Behre, Jr., professor, geology dept., Columbia University.

(37) *Geologic Factors Leading to Negative Results in Exploration at White Pine, Tennessee.* Arnold Brokaw.

(38) *Clay Alterations Associated with Uranium Mineralization.* Paul F. Kerr, professor of mineralogy, Columbia University.

Exploration Costs. Joint session MGGD and MED.

For list of papers see MED. Tuesday, A.M.

General Session organized by John J. Collins.

(39) *Interrelation of Mining Geology and Mineral Dressing.* F. W. Farwell and Donald W. Davis.

(40) *Mine Drainage Studies in the Iron Ranges of Northern Michigan.* Wilbur T. Stout.

(41) *Origin of the Mississippi Valley Type Lead-Zinc Deposits—A New Hypothesis Extended.* A. F. Frederickson, associate professor of geology, Washington University.

(42) *Geology of Hayden Creek Lead Mine, Southeast Missouri.* Ernest L. Ohle, geologist, St. Joseph Lead Co.

Clay and Laterite Relationships. Joint session with SEG and Ind. Min. Div. Organized by A. F. Frederickson, associate professor of geology, Washington University.

(43) *Origin of Arkansas Bauxite Deposits, Part 1—On Field Relationships. Part 2—Processes That Formed the Deposits.* M. Gordon and J. Tracy.

(44) *Occurrence and Exploration of Kaolin Deposits in Central Georgia.* Thomas L. Kesler, Thompson, Weinman & Co.

(45) *Observations on the Relations of Hydrous Aluminum Oxide Minerals to Clay.* V. T. Allen.

(46) *Recent Advances in Clay Mineral Technology.* R. E. Grim.

(47) *Iron Oxides and Hydroxides in Laterites.* R. Mackenzie.

GEOPHYSICS SUBDIVISION

General Session, organized by H. W. Straley, III, Georgia Institute of Technology, Atlanta, Ga.

(48) *Annual Review of Geophysics.* Sherwin F. Kelly, president, Sherwin F. Kelly Geophysical Services, Inc.

(49) *Determination of the Temperature and Pressure of Formation of Minerals by the Decrepitometric Method.* F. G. Smith, University of Toronto.

(50) *Low Temperature Mineralization in the Canadian Shield and its Relation to Gold Deposition.* W. M. Little.

(51) *Variations in the Thermo-electric Properties of Pyrite in Association with Gold Ore.* A. D. Mutch.

(52) *Airborne Magnetometric Exploration in the Petroleum Industry.* Homer Jansen.

(53) *Contribution of Geophysical Surveys to the Discovery of Stilfontein Gold Mine in South Africa.* Oscar Weiss, consulting geophysicist, Weiss Geophysical Corp.

(54) *Application of Geophysical Methods of Underground Exploration in Civil Engineering.* George F. Sowers.

(55) *Practical Results from Resistivity Methods as Applied to Ground-water Problems in Arizona.* Samuel Turner.

(56) *Case Histories of Resistivity Applied to Near-Surface Geology.* William W. Schwendiger and Walter J. Gabriel.

(57) *Electrical Resistivity Measurements at the Meramec Plant Site, St. Louis County, Missouri.* LeRoy Scharon, professor, geophysics, Washington University.

Donations for Books to Belgium Library Solicited

While the famous chimes given to the library of the University of Louvain, Belgium, by a group of American engineering societies after World War I, survived the fire and bombing of World War II, the books in the library were destroyed. The building has been restored, shelves have been replaced, and some books have been collected. But few recent books in English fill those empty shelves.

A plea has been made to American engineers to help in this matter. The CARE UNESCO Book Fund will channel any donations for books to Louvain; and a bookplate inscribed with the name of the company or organization will be inserted on all gifts of $10 or more. While Belgium and Europe were in a long war, resulting in an intellectual blackout, American research and progress continued. Recovery in this area can be speeded by making American knowledge available through subscription to Louvain through the CARE UNESCO Book Fund.

Appoint EJC Nominees To Science Foundation Board

Donald H. McLaughlin, AIME President; Edward L. Moreland, principal in consulting firm of Jackson & Moreland; and Andrey A. Potter, dean of engineering, Purdue University, were recently named by President Truman as members of the new 24-man Board of National Science Foundation. These three engineers were among the nominees presented to President Truman by Engineers Joint Council. With the EJC list of nominees went a strong recommendation that applied science, as represented by engineers, be given appropriate consideration on the Board of National Science Foundation.

AIME Receives Two Grants

Two substantial gifts were received by the AIME Board of Directors at its meeting on Nov. 15. One was a contribution of $5000 from the Cities Service Co., through the Doherty Memorial Fund, for special services to Petroleum Branch members. Authorization has been given to use income from the Doherty Fund to underwrite a volume on petroleum conservation, the manuscript which has just been completed by Stuart Buckley, and to offset a part of the cost of publishing petroleum statistics. The same amount also has been contributed in other recent years.

A gift of $2250 was made by the American Smelting & Refining Co., through the Seeley W. Mudd Memorial Fund, for the purchase of 1500 copies of the volume entitled Metal Magic. This will be distributed free of charge to new Junior Members of the AIME in the coming year or two.

MIED Annual Meeting Planned

The 1951 annual meeting of the Mineral Industry Education Div. of AIME will be held at 2:30 pm on Sunday, Feb. 18, 1951 at the Jefferson Hotel in St. Louis. The afternoon and evening forums will contain features of interest in mining, milling, metallurgy, geology, geophysics, and petroleum engineering. A buffet supper will be served in the interval between afternoon and evening sessions. Members planning to attend should notify dean Carl Tolman, Graduate School of Arts & Sciences, Washington University, St. Louis.

Mexico Section Welcomes Visitors

Luncheon meetings of the newly organized Mexico Section of the AIME are held on the first Monday of each month at the American Club in Mexico City. AIME members who may be in the city at that time will be welcomed. They should get in touch with William G. Kane, Secretary-Treasurer of the new Section, whose Post Office address is Apartado 711, Mexico, D. F., and whose residence is San Juan de Letran No. 9, Desp. 805; telephone 10-27-24.

Officers of the new New Mexico Section are: Chairman, Alfredo Terrazas; Vice-Chairman, Valentin Garfias; Secretary-Treasurer, William G. Kane.

Student Associate Arrearages Eliminated

Several actions were taken by the AIME Board of Directors at its meeting on Nov. 15 to benefit Student Associates. Instead of continuing Student Associates on the rolls after they become in arrears for dues they will be dropped at the end of three months after their dues are payable. Thus they can apply for reinstatement and change of membership grade at any time in the future without being penalized for any arrears in dues that may have been incurred. Where such arrears have been debited against a Student Associate in the past, the sum of $1 is to be charged for reinstatement, regardless of the amount of accumulated arrears. Student Associates are also exempted from any registration fee at the forthcoming annual meeting in St. Louis, though they will receive the same technical papers as are distributed to those who pay the $5 or $8 registration fee. Students who are not Student Associates of the Institute will pay a registration fee of $1.

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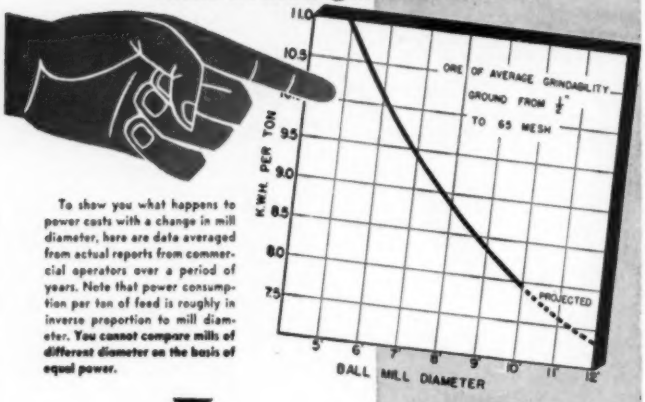

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Subjects Discussed At Annual Fuels Conference

The 13th Annual Joint Fuels Conference of the Coal Div. AIME and the Fuels Div. ASME was held at the Hotel Statler, Cleveland, Oct. 24 to 25, the Cleveland Engineering Society cooperating.

H. A. Baldwin presented a paper on the causes and probable effects of today's fuel situation. He indicated that the coal industry has done little to develop customer interest in the past 20 years and suggested that better public relations could help solve the problems of competition. B. A. Landry pointed out that the industry started on research 15 years ago, and he cited examples to show that progress had been made.

J. B. Morrow suggested that at some future time the gas pipe lines crisscrossing the country would come in handy to transport producer gas. Elmer Kaiser said that coal had had to compete with wood, water power, coal from England, and animal power. The best facilities for producing coal are in this country. As for making gas and motor fuel out of coal, there are a number of possibilities.

Marshall Pease, Jr., and R. J. Brandon described a pattern for sound utility fuel procurement. Mr. Brandon outlined the method used in selecting and evaluating fuel to be burned and presented data which have been accumulated over the past 10 years. Mr. Pease concluded by outlining the four problems facing the coal industry: (1) Lowering operating costs by mechanical methods while maintaining low ash and low water content, (2) Handling coal during the winter, (3) Labor relations, and (4) Government controls. It was brought out that some small coal operators are mistakenly trying to improve preparation when actually they are increasing the water content more than the ash content is decreased.

B. A. Landry and A. L. Bailey in their paper on the effect of coal size on the sampling of coal for float-and-sink tests discussed the problem of reduction in weight of a gross sample of coal by subdivision to a convenient weight for laboratory handling without loss of accuracy.

Cleveland industries spent more than $4 million on smoke and air pollution abatement in 1949 without prosecution or persecution, according to H. G. Dyktor, commissioner, Cleveland Div. of Air Pollution, who spoke at the Tuesday luncheon; his subject, "Meet the Smoke Inspector."

At the banquet Tuesday evening, the Percy Nichols Award was presented in behalf of the two divisions to Julian E. Tobey, president, Appalachian Coals, Inc., by E. G. Bailey.

At a joint meeting with the executive committee, Fuels Div., ASME, it was agreed that the 1951 Fuels Conference will be at the Roanoke Hotel, Roanoke, Va., Oct. 11 to 12, and the 1952 conference at Philadelphia.

Reported by E. J. Kennedy, Jr.

Drift of Things . . . . . as followed by **Edward H. Robie**

DEMAND and supply of engineers, present and future, has had more than the usual amount of attention in the past year. Since the end of the last World War, new peaks have been attained in engineering enrollments in the colleges. The profession seemed to be particularly attractive to the GI's who took advantage of the Government's offer to finance their educations. In the prewar decade, some 11,000 engineers had been graduated annually. The figure rose to 19,000 in the academic year 1946-47, to 31,000 in 1947-48, to 47,000 in 1948-49, and to 51,000 in 1949-50. This for a time saturated the market, and those graduating last June found a buyer's market.

A year or more ago the Dept. of Labor came out with a flat statement that *"the number of graduates will greatly exceed the number of engineering job openings in the next few years."* This was based on a study that indicated that the number of graduates needed would approximate 17,000 or 18,000 annually for the next several years. Contrasted with this, it was estimated that 36,000 would be graduated in 1951 and 29,000 in 1952. Unfortunately this pessimistic statement received wide publicity and discouraged many young men from entering engineering. Difficulties reported in obtaining jobs six months ago added to their feeling that perhaps for the next few years engineering was a poor field to enter. Freshmen engineering enrollment currently is only about 27,000. Instead of more than 5 pct of high school graduates going into engineering, as in 1947, this year only 2.2 pct selected this field.

Selective Service has indicated that the upper half of the current 27,000 students will be deferred to continue their education next year, or 13,500. Those in the upper two thirds of the sophomore class, or 9000, could continue as juniors; and the upper three quarters of the juniors, or perhaps some 7000, could graduate in 1954. Against this, Dean Hollister, of Cornell, believes 20,000 engineering graduates will be needed annually for peacetime needs alone, and that the present emergency will increase this number, for both civilian and military needs, to 30,000.

If Selective Service were not withdrawing any engineering students at all, Dean Hollister estimates the number of graduates in 1951 at 32,500; in 1952 at 21,900; in 1953 at 17,000; and in 1954 at 12,400.

It is plain that the top half of high school students, particularly those who rank high in mathematics, science, and may we add English, should be apprised of the excellent opportunities that seem almost certain to be open to engineering graduates in the next few years. AIME members can help to do this, and present engineering students can tell their younger brothers and their friends that this is a field of high promise.

Furthermore, those who feel no particular urge to go out and shoot somebody, and have the intellect to stay in the top half of their class at an engineering college, have some assurance that they will be allowed to serve their country in a more constructive way than by carrying a gun. Engineers Joint Council, of which the AIME is a component society, is doing everything possible, through its Engineering Manpower Commission, to see that an adequate flow of engineers shall come out of our schools in the future years, and that their talents are thereafter used to the best advantage.

Misery Loves Company

Our good friend Austin Wright, general secretary of The Engineering Institute of Canada, writes: "You will have no idea of the kick that I got out of your item in the 'Drift of Things' in the November issue. Your experience with your directory was so much like ours that it really did me a lot of good to read your account. Change the name of the organization and your article might have been written by some one in the Engineering Institute. However, I hope that by the time you get through with it all you will not be out as much money as we were on the deal. It cost us $18,000 to print 15,000 copies."

Well, *our* printer's bill, for which we have just reluctantly signed a check, was for $15,999.11, for 16,000 copies. (We had twice as many pages as the Canadian volume but they were only half as large.) In addition we charged some $3000 against the volume for salaries, postage, mailing expense, post cards circulated to members, and aspirin for all concerned.

Book Manuscripts Wanted

Twenty years ago the family of the late Seeley W. Mudd contributed $100,000 to the AIME, establishing a fund whose income was to be used *"for the advancement of the sciences of mining and metallurgy by the encouragement of research and the dissemination of knowledge, and for the promotion of the welfare of engineers engaged in the professions of mining and metallurgy."* A further suggestion was that preference be given to projects of particular benefit to young engineers. Accumulated unspent income in the fund now totals some $36,000. A part of the income heretofore has been used to publish or purchase books, subsequently given free to new Junior Members of the Institute. Money in the fund also has been used to underwrite preparation and publication of such books as: *"Industrial Minerals and Rocks," "Elements of the Petroleum Industry," "Coal Preparation," "Basic Open Hearth Steelmaking,"* and *"Biringuccio's 'Pirotechnia'."*

The Committee administering the fund is now seeking material for new volumes, either for free distribution to Junior Members or, in the latter category, for general sale. If any member of AIME has the material for a good book in his system, or knows of some one who has; or if he can suggest a topic and an author for a book that will fill a gap in the literature; then we should like to be advised.

New Edition of Agricola Popular

As this is written, early in December, we received orders for more than 500 copies of the reprint of the Hoover translation of Agricola's *"De Re Metallica."* This represents but four days' mail, following our circular to members. The total edition of the reprint was 1000 copies, of which half had been reserved for us. The response exceeded our expectations greatly. We shall do what we can to get more copies, but the demand may well amount to 1000 copies from AIME members alone.

For Thirsty Members Abroad

Stanley Lefond sends us the accompanying picture from Sfax, Tunisia. He says that he knew the AIME was not in too good shape financially, but he did not know that we had to resort to such measures as this.

Personals

J. L. Alexander is now a mine assistant for the New Guinea Gold Fields, Ltd., Wau, New Guinea.

Dionys Burger has resigned from the geological staff of Mount Isa Mines Ltd., Mount Isa, Queensland, Australia. He has been appointed geologist, State Dept. of Mines, Tasmania.

John F. Barkley, chief of the fuel utilization branch of the fuels and explosives div. of the Bureau of Mines, was recently elected a Fellow in the ASME in recognition of his outstanding professional record on improving fuel efficiency and the economical operation of government power and heating plants.

C. A. Botsford, mining consultant at Rome for the ECA Mission to Italy has been transferred to the ECA Mission to Greece at Athens.

Wen-Lan Chen recently received his Ph.D. in mining engineering from the Pennsylvania State College and is an assistant professor at Oregon State College, Corvallis.

W. F. Camacho-Navarro is now employed by Basic Refractories, Inc., Maple Grove, Ohio. He spent several months in Ecuador following his resignation from the explosives dept. of the American Cyanamid Co.

Frank W. Cole, Humble Oil & Refining Co., Houston, was recently transferred from Tomball, Texas as a junior petroleum engineer.

L. I. Cothern is director of engineering for the Jewell Ridge Coal Corp., Tazewell, Va.

Richard V. Colligan has been transferred from superintendent of sulphur exploration for the Freeport Sulphur Co., New Orleans, to the New York office as resident engineer.

C. Arthur Carlson has accepted the position of manager with the Knight Ideal Coal Co., Price, Utah. He was previously associated with the Spring Canyon Coal Co., Spring Canyon, Utah.

A. B. Crichton, president of the Johnstown Coal & Coke Co. was honored by his fellow citizens in Johnstown by a testimonial dinner given as part of the city's Sesqui-Centennial Observance. Mr. Crichton was cited for being "an integral factor in the development of the local soft coal area" for the past 50 years, as well as "a man of tremendous energy always striving to advance the best interests of his various enterprises and his community." Various notable figures, local, state, and national, were on hand to pay tribute to Mr. Crichton.

Simon F. Coats, Jr. is representing the Joy Mfg. Co. in Ankara, Turkey.

William G. Fountain, a recent graduate of Michigan College of Mining & Technology, Houghton, Mich. is now a member of the Gogebic range engineering staff of Pickands Mather & Co. at Ironwood, Mich.

James E. Forciea is now a laboratory assistant for the Oliver Iron Mining Co., Duluth, Minn.

S. P. Gupta is now taking underground practical training in the collieries of M/S Macneill & Barvy, Ltd., Burdwan, India.

Milton B. Harper has joined the Island Creek Coal Co., Holden, W. Va. as section inspector.

J. D. Clendenin has been named to head the preparation and utilization branch of the U. S. Bureau of Mines at its new anthracite research laboratory at Schuylkill Haven, Pa.

J. D. Clendenin

K. K. Hood has been assigned to supervise the Eagle Valley zinc mine at Pioche, Nev. for the Callahan Zinc-Lead Co. Until recently he was in charge of a mine in Colorado for the same company.

H. R. Hendricks, formerly mill superintendent at Bisbee, Ariz., for the Shattuck-Denn Mining Corp., is now with the Buchans Mining Co., Ltd., Buchans, Newfoundland.

Oliver Hall, mining and consulting engineer, has been elected a director of Noranda Mines, Ltd., Toronto, Ont.

John M. Hedin has joined the engineering department of Pickands Mather & Co., Ironwood, Mich.

Robert C. Hills has been elected a vice-president of the Freeport Sulphur Co., New York. He was formerly director of development for the same concern.

R. P. Ietter is now sales manager, overseas distributors, for the Joy Mfg. Co., Paris. He was formerly French liaison officer with SHAEF G-4 solid fuels.

Rudolph Krahmann, resident of South Africa who became a Life Member in 1934, recently contacted Institute headquarters after an 11-year period during which he had been a victim of the fortunes of world war. In September 1939 he was interned by South African authorities as a German national. Five years later he was released to Germany under a civil internees exchange plan. After the war's end and from May 1945 to August 1948, he was detained by the Russian Secret Service as a prisoner without trial, and only recently returned to South Africa. His present address is 36 Trysa Bldg., Pretoria, Transvaal.

Myron C. Kiess has joined the Mid-Continent Petroleum Corp., Tulsa. He was previously associated with the Pure Oil Co., Houston.

H. A. Kursell, consulting engineer for the American Smelting & Refining Co., New York, is now in South America.

J. C. Kinnear, Jr., has been appointed general manager of the Nevada operations, Ruth, Nev., of the Kennecott Copper Corp., succeeding **Walter Larsh** who is retiring. **Paul Hett,** general superintendent has been promoted to assistant general manager of Kennecott's Nevada copper mining operations.

Edmond A. Krohn has accepted a position with the U. S. Steel Corp., Dragerton, Utah. He is associated with the engineering staff of the coal mines.

M. H. Kuryla is now director of the U. S. Smelting, Refining & Mining Co., Boston.

James L. Head, Anaconda Copper Mining Co., New York has been elected president of the Alumni Assn. of the Missouri School of Mines, Rolla, Mo. Mr. Head graduated with the class of 1916.

James L. Head

D. P. Lambert has joined the Canadian Copco Ltd., Montreal A.M.F., Que.

C. E. Lesher has announced his retirement as president from the Pittsburgh Consolidation Coal Co. and its predecessor, Pittsburgh Coal Co., now the Disco Co. He plans to engage in consulting practice as a specialist in coal processing and the economics of production and utilization of coal and its products.

Orville R. Lyons, chairman of the Program Committee of the Coal Div., Mining Branch, AIME, has been made manager of preparation, Republic Steel Corp., Cleveland.

A. E. Marshall has opened his office as a consulting chemical engineer in the Industrial Trust Bldg., Providence, R. I. He was recently vice-president in charge of the Rumford Div., Heyden Chemical Corp.

S. J. McCarroll, a consulting engineer, is now located at 1446-12th, Douglas, Ariz.

Stanley A. Mayer has accepted the position of superintendent of the Ticlio Div., Volcan Mines Co., Ticlio, Peru. For the past five years he was in Bolivia for the Patino interests as mine superintendent and similar positions.

Padraic Partridge, mining engineer and geologist for the Filtrol Corp., has been transferred from the Los Angeles office to the Salt Lake City office.

P. H. Reagan, consulting mining engineer of Hye, Texas, returned to the United States after doing professional work in Australia. He will make a short trip to Latin America before returning to Australia in March.

Arnold H. Miller, consulting engineer of New York, left for Europe where he will be engaged in professional work. He will go to Spain, French Morocco, Italy, France, Western Germany, and England.

Arnold H. Miller

Vincent H. Ream is presently employed as production engineer with the Olga Coal Co., Coalwood, W. Va.

John W. Owen is now assistant mill foreman for the Cerro de Pasco Copper Corp., Mahr Tunel, Peru.

Howard H. Rice of Butte, Mont., has been awarded a $500 scholarship for the 1950-51 scholastic year presented by the American Smelting & Refining Co. Mr. Rice is a senior majoring in mining engineering at the Montana School of Mines.

David E. Morgan has returned to this country after a seven-month trip to England, Belgium, France, Tunisia, Morocco, and Sardegna, where he visited the coal and metal mines. Mr. Morgan is president of the Peerless Precision Products Co., Providence, R. I. and vice-president of the International Mfg. & Equipment Co., Inc., New York. He is also representing major American mining equipment manufacturers. After a short stay in this country, he contemplates an extensive trip to the Middle and Far Eastern countries where he will operate a coal and a chrome mine.

David E. Morgan

Pierre Routhier, chief of the geological laboratory, University of Paris is now engaged in the preparation of a geological map of New Caledonia and the publication of his work on the geology and mineral resources of this island.

Glenister Sheil is now general manager of the Mount Morgan, Ltd., Mount Morgan, Queensland, Australia.

Charles A. J. Schulte of Vanadian Corp. of America has been transferred to the Niagara Div. at Niagara Falls, N. Y.

Frank J. Stortz is on leave from the Bureau of Mines to advise the authorities of Colombia, S. A. on the opening and development of their coal deposits required for the Paz de Rio steel plant project located approximately 150 miles north of Bogota.

Frank J. Smith

Frank J. Smith has been elected vice-president in charge of iron ore operations of the Oglebay, Norton & Co., Cleveland. He will continue with the same responsibilities he formerly had in his position of range manager. He will be located temporarily in Montreal, Wis.

Curtis F. Rivers has opened his own company, the Rivers Equipment Sales, Birmingham, and is engaged in selling used mine and industrial equipment.

Stuart St. Clair, consulting engineer of New York, recently returned from a two months mineral and economic survey throughout Indonesia. He was accompanied by **Frank M. Estes** of Memphis.

Francis A. Stejer, U. S. Geological Survey, Alaskan section, completed field work on the Berners Bay portion of the Juneau Gold Belt. He has returned to Washington, D. C.

R. Harold Stacy is now employed by Sloss-Sheffield Steel & Iron Co., Birmingham working in coal preparation. He was previously associated with the H. C. Frick Coke Co., Uniontown, Pa., as research assistant.

C. DeWitt Smith has been promoted to superintendent of the Balmat and Edwards zinc mines in St. Lawrence county, New York of the St. Joseph Lead Co. He was formerly mining captain of one of the lead mines in Missouri for the same company.

Carlton T. Spalding, former party chief and seismologist for Rogers-Ray, Inc., Houston is now connected with the General Petroleum Corp., Great Falls, Mont. as a geologist.

Pieter B. Vogelsang is now employed by the Devon Corp., La Paz, Bolivia.

James H. Wren is manager of J. H. Wren & Co., consulting engineers, 4297-D St., Sacramento. This was an incorrect listing in the directory.

Robert G. Woods is now employed as a chemist by the Yucca Mining & Milling Co., Yucca, Ariz. He was previously connected with San Luis Mining Co., Tayoltita, Durango, Mexico.

Obituaries

Fred A. Austin (Member 1944), mining engineer, died in March 1950. Born at Grass Valley, Calif. in 1880, he received his engineering education at various schools. He concluded his education at Van der Naillen School of Engineering, graduating in 1907 with the degree of M.E. For approximately 14 years he was employed by various concerns working at mining and milling of metalliferous ores and doing geological work. In 1924 he joined the Original Sixteen to One Mine, Inc., Alleghany, Calif. as a mining engineer and did geological work. In 1943 he began his private practice as a mining engineer.

Charles Volney Averill (Member 1940) died on Aug. 28, 1950 after a brief illness. Mr. Averill was mining engineer for the California State Div. of Mines for more than 20 years. He was born on June 10, 1892 at Virginia City, Nevada and graduated from the College of Mines, University of California, Berkeley in 1914 with a B.S. degree. Following his graduation he was employed as a miner, mechanic, assistant surveyor and geologist for the West End Mining Co., Halifax Mining Co., and Tonopah Belmont Development Co. During the following years prior to joining the State Div. of Mines in 1928, he worked in various phases of mining and geological work. He worked at the Redding, Sacramento, and San Francisco offices before being made supervising mining engineer in 1947. He wrote articles on nearly all phases of mining and geological activity in California.

Paul W. Avery (Member 1918) died on Sept. 27, 1950. He matriculated at Stanford University, and received his A.B. in 1905 and his masters degree in 1906. His first position was as assistant chemist for the Homestake Mining Co., Lead, S. Dak. He went to Mexico in 1909 as chemist and assayer for the Esperanza Mining Co. In 1911 he went to Nicaragua as metallurgist and assistant general manager for the La Leonesa Mine, Ltd. He remained in Mexico until Sept. 1946 when failing health caused his retirement and he returned to California.

Ralph S. Blitz (Member 1939) died on May 28, 1950. Born in Minneapolis in 1884, he attended the University of Minnesota from 1902 to 1904 and the University of Idaho from 1904 to 1905. For three years he was employed as a mucker, miner, timberman, and did other various work in the mining field. In 1908 he joined the Pittsburg Silver Peak Gold Mining Co., Blair, Nev. as an engineer. For the next seven years he was employed by the Davis Daly Copper Co., Butte, and in 1917 became manager of the Philipsburg Mining Co., Philipsburg, Mont. Following six years as manager for the U. S. Vanadium Corp., Rifle, Colo. he entered private practice for approximately one year. He joined the Denver Equipment Co., Denver, as engineer in 1932 and in 1937 became superintendent for Gilt Edge Mines, Inc., Deadwood, S. Dak. In 1941 he was manager of the Naturita Div., Vanadium Corp. of America, Naturita, Colo. A few years later he again opened his office as a consulting engineer.

Walter E. Burlingame (Member 1940), mining engineer, died on Aug. 10, 1950. He was born in 1879 at Denver and graduated from the Colorado School of Mines in 1901 with the degree of Engineer of Mines. For approximately 20 years he was in charge of his own business in assay, chemical and metallurgical testing. In 1923 he joined the New Life M. & M. Corp., Rollinsville, Colo. as manager. He was employed by the Cross Gold Mining Co. in 1933 doing metallurgical testing work. He did examining and consulting engineering work from 1933 to date.

Edwin Carl Carlson (Member 1942) died on Sept. 9, 1950. Mr. Carlson was born at Ironwood, Mich. in 1893 and in 1926 received his B.S. & E.M. degrees from the Michigan College of Mines. Following graduation he was employed as a mining engineer for the Oliver Iron Mining Co., Ironwood, Mich., at the Norrie mine. In 1927 he joined the Youngstown Mines Corp., Ironwood at the Anvil-Palms mine. For three years he was connected with the Sunday Lake Iron Co., Ironwood as a mining engineer. He returned to the Youngstown Mines Corp. in 1931 as assistant superintendent.

Willis H. Carrier (Member 1938), founder of the Carrier Corp., died on Oct. 7, 1950. Born in Angola, N. Y. on Nov. 26, 1876, Dr. Carrier attended Cornell University and graduated in 1901 with the degree of M.E. Following graduation he joined the Buffalo Forge Co. as a research engineer and in 1906 became chief engineer. While employed by this company his interest in air conditioning had its origin and he developed what was known as a spray-type air washer. In 1905 he devised a spray-type air conditioner capable of heating, cooling, humidifying air, an apparatus that provided the basis for those which are on the market today. As a result of this work he formed the Carrier Engineering Corp., Newark in 1915, which later was known as the Carrier Corp. Dr. Carrier served as president until 1931, becoming chairman of the board until 1943. In 1948 he was named chairman-emeritus. He presented a paper in 1911 which was ranked as fundamental engineering doctrine and upon which was built the science of air conditioning. In the ensuing years he made numerous contributions to air conditioning.

Cecil Oliver Dunlop (Member 1941), president and manager of the Silver Dollar Mining Co., Wallace, Idaho, died on Aug. 10, 1950. He was born at Minneapolis in 1890 and attended public schools. In 1929 he was assistant manager of the Stratton Silver Summit, Wallace, Idaho. He became president and manager of the Silver Dollar Mining Co. in 1933 and remained with that concern until the time of his death.

Charles Arthur Franks (Student 1948), a former student at the New Mexico School of Mines died. Born in Manchester, N. H. in Apr. 1929, he had expected to graduate in 1951 with the degree of B.S.

Orr R. Hamilton (Member 1917) died on Apr. 24, 1950. Mr. Hamilton attended school at Newark, Ohio where he was born in 1882, and matriculated at Michigan College of Mines in 1906 receiving his E.M. After graduation he joined the Cerro de Pasco Mining Co. in Peru as an engineer. In 1907 he became associated with Sociedad de Minas de Cobre de Cutter Cove

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also in South America. He came back to the United States in 1909 and was employed as an engineer for Pickands Mather Co. and several concerns in Michigan.

Robert K. Matheson (Member 1939), mining engineer, died on Aug. 27, 1950. At the time of his death he was employed as night superintendent, U. S. Potash Co., Carlsbad, N. Mex. He was born in Phoenix, B. C. in 1908 and attended the University of British Columbia, receiving his B.A.Sc. in 1935. Following graduation he was employed by various companies in Canada and in 1937 became chief engineer for Desert Silver, Inc., Silver Peak, Nev. In 1938 he was made mine foreman for the same concern. He spent four years in the Royal Canadian Air Force and after his discharge in 1946 he joined the Cordero Mining Co., McDermitt, Nev. as general superintendent. After two years he became general superintendent for the Corondao Copper & Zinc Co., Bella Vista, Calif.

Charles Morgan (Member 1941) died on May 20, 1950. He was born in Jumbrota, Minn. on Feb. 12, 1880. In 1906 he graduated from the University of Minnesota School of Mines and received his E.M. After graduation he worked on the Mesabi range at Hibbing for several years. He was sent to Alabama by the Republic Steel Corp. and made superintendent of Woodward Iron Co.'s brown ore mines. Mr. Morgan did extensive geological work with the Alabama Geological Survey and the U. S. Geological Survey. For five years he was supervising mining engineer for the Reconstruction Finance Corp., Birmingham. At the time of his death he was engaged in private mining and consulting work.

Claude Ervin Needham (Member 1943), mineral economist, died on Oct. 15, 1950. Dr. Needham was well-known throughout the mineral industry for his work in sedimentary petrography, stratigraphy, micropaleontology, and mineral economics. He was born at Newton, Ill. on Sept. 10, 1894. He attended Central Normal College at Danville, Ill., Iowa State College, and received his B.S. degree in 1924 and M.S. degree in 1925 from Mississippi State College. In 1931 he earned his Ph.D. at Northwestern University. Dr. Needham was appointed assistant professor of geology at Mississippi State College in 1926 and an instructor in geology at Northwestern University in 1927. In 1931 he went to Socorro, N. Mex. as assistant professor of geology at the New Mexico School of Mines and became president of that institution in 1939. He served in that capacity until 1942 during which time he was also director of the New Mexico Bureau of Mines and Mineral Resources. He joined the U. S. Bureau of Mines as senior economic analyst in Washington, D. C. in 1942. Later he served for five years as supervising engineer of the Economics and Statistics Branch at Salt Lake City. In 1950 he was appointed chief of the Mineral Statistics Branch, Region VI of the Bureau of Mines with headquarters in Amarillo, Texas. He contributed technical articles to professional journals, Bureau of Mines publications, and from 1943 to 1945 was editor of the Minerals Yearbook.

Necrology

Date Elected	Name	Date of Death
1950	C. P. Anderson	Unknown
1941	Lewis Edgar Garfield	Unknown
1917	John C. Gaul	Oct. 23, 1950
1943	Irvin Gladstone	May 9, 1950
1921	Arthur Crandell Green	Oct. 31, 1950
1945	Joseph John Haberthier	Aug. 27, 1950
1947	John Robert Hallock	Oct. 15, 1950
1937	George H. Harbordt	Oct. 19, 1950
1913	Charles W. Johnston	Oct. 24, 1950
1926	Virgil R. D. Kirkham	Unknown
1902	William B. McKinley	Oct. 21, 1950
1941	Charles Morgan	May 20, 1950
1936	C. Quinby Schlereth	Nov. 10, 1950
1921	Temple W. Tutwiler	Sept. 9, 1950
1943	M. Harley Wilcox	March, 1950
1900	Allen H. Woodward	Nov. 26, 1950

— Coming Events —

Jan. 3, AIME, Chicago Section, Chicago Bar Assn., Chicago.

Jan. 8, AIME, Colorado Section, Denver.

Jan. 8-12, SAE, annual meeting, Hotel Book-Cadillac, Detroit.

Jan. 9, Society for Applied Spectroscopy, 6 pm dinner, Tosca's; 8 pm meeting, Socony-Vacuum Training Center, New York.

Jan. 15-17, 1951, AIME, Minnesota Section, annual meeting. Mining symposium conducted by Center for Continuation Study, University of Minnesota.

Jan. 18-20, Society of Plastics Engineers, annual national technical conference, Hotel Statler, New York.

Jan. 23-26, International Heating & Ventilating Exposition, Commercial Museum, Philadelphia.

Feb. 18-22, AIME, annual meeting, Jefferson Hotel, St. Louis. Metals Branch session to be held at the Statler Hotel.

Feb. 21, AIME, Western Section, Open Hearth Committee, Iron and Steel Div., Los Angeles.

Mar. 5-9, ASTM, spring meeting and committee week, Cincinnati.

Mar. 11-14, American Institute of Chemical Engineers, regional meeting, Greenbrier, White Sulphur Springs, Va.

Mar. 13-16, National Assn. of Corrosion Engineers, conference and exhibition, Hotel Statler, New York.

Apr. 2-4, 1951, AIME, Open Hearth and Blast Furnace, Coke Oven and Raw Materials Conference, Iron and Steel Div., Statler Hotel, Cleveland.

Apr. 2-5, ASME, spring meeting, Atlanta-Biltmore, Atlanta.

Apr. 16-18, American Society of Lubrication Engineers, national convention, Bellevue-Stratford Hotel, Philadelphia.

Apr. 23-26, American Foundrymen's Society, annual convention, Buffalo, N. Y.

Apr. 25-26, Metal Powder Assn., annual meeting, Hotel Cleveland, Cleveland.

Apr. 30-May 11, British Industries Fair, annual industrial show, Olympia and Earls Court, London; Castle Bromwich, Birmingham.

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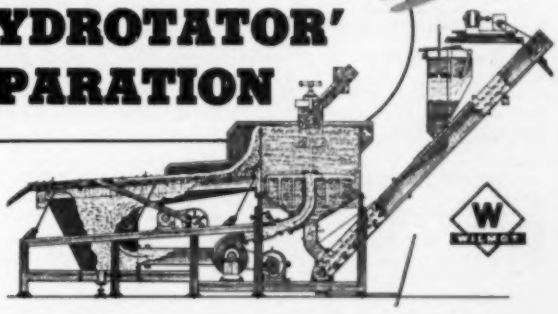

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In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; AM, Associate Member, S, Student Associate.

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Phoenix—Reynolds, Robert L. (J) (C/S—S-J)
Prescott—Pessin, Albert L. (M) (C/S—J-M)
Prescott—Still, Arthur R. (J) (R. C/S—S-J)
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New York—Waterman, Glenn C. (M) (R. C/S—J-M)

(Continued on page 80)

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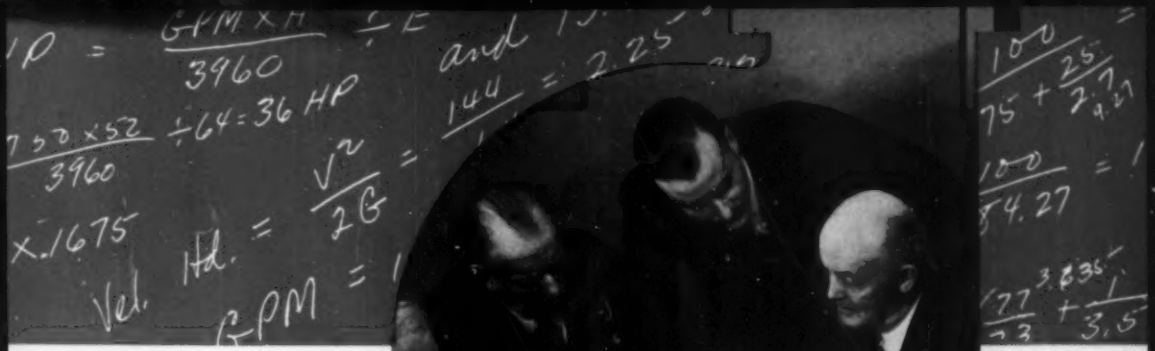

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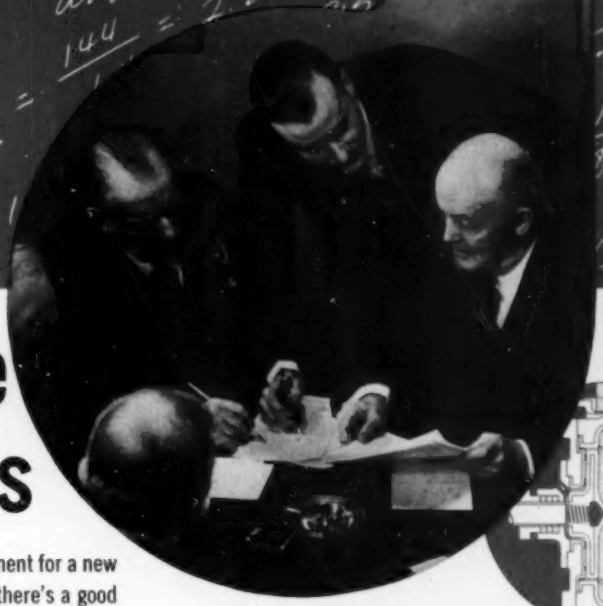

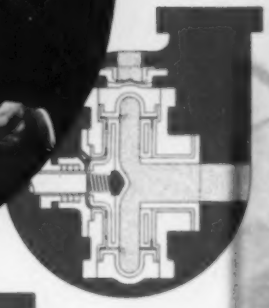

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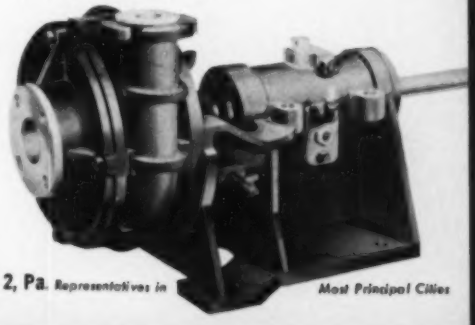

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